AMCL Associated Mining Consultants Ltd.
P.O. Box 6500, Station D, 840 - 7 Avenue S.W., Calgary, Alberta T2P 2V8
Tel: (403)264-9496 Telex: 038-22761

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Department of Energy, Mines and Resources
CANMET, Coal Research Laboratories
210 George Street
SYDNEY, Nova Scotia
B1P 1J3

Attention: Mr. D.B. Stewart, P.Eng.
Scientific Authority

Dear Sirs:

Re: Underground Coal Mine Fire Control with
Inerting Systems - A Study
DSS File # 03SQ.23440-3-9035
DSS Contract Serial No. 0SQ83-00155

We take pleasure in submitting our final report on the control of underground coal mine fires using inerting systems.

We would like to thank the Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources, Canada, for sponsoring the study and for their advice and guidance during its execution.

Respectfully submitted,

ASSOCIATED MINING CONSULTANTS LTD.

Alan L Craven,

Alan L. Craven, P.Eng.
Manager

ALC:ms

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UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

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Westar Mining Limited, CANADA

STUDY TEAM

This study was carried out by:

John P.L. Bacharach, P.Eng.
Gordon V. Jolliffe, C.Eng.
Alan L. Craven, P.Eng.

assisted by the staff of Associated Mining Consultants Ltd.

EXECUTIVE SUMMARY

Inerting is the process of introducing inert gas into the atmosphere in contact with a fire to reduce the oxygen level and hence to control the intensity of the fire to the point of extinction. Inerting is not a substitute for conventional fire control methods, but is a valuable addition to the range of techniques that can be applied to combat underground mine fires. The benefits of inerting include reduced explosion hazard, reduced fire intensity, reduced air leakage and easier recovery of the fire zone.

The report includes a section that provides a general introduction to mine fires, causes, their nature and effects. Fire fighting procedures and precautions are reviewed.

The report describes the current status of mine fire control using inerting techniques from the perspective of the Canadian coal mining industry. These techniques have been used in many countries over a long period of time and are now well established in the major coal industries of the world. Many notable successes have been achieved and the use of inerting is increasing both in western and eastern bloc countries.

Several types of fire are possible in Canadian underground mines. These include open fires in roadways and other accessible locations on longwall faces, or in gob areas, heatings in rib sides, pillars or other solid coal areas and in mined-out areas. Inerting techniques appropriate to Canadian circumstances must, therefore, be capable of being used to combat open fires and to prevent or control heatings.

Availability, cost and other factors have focused the coal industry's attention on three inerting media - gaseous combustion products, carbon dioxide and nitrogen. Sufficient success has been achieved with these media in many parts of the world to provide a high degree of confidence that one or more will be appropriate for Canadian conditions. The relative merits of these three inerting media and their respective delivery systems have been reviewed and ranked according to their suitability for application in Canadian underground coal mines. The inerting technique recommended for general application in the Canadian coal industry is the use of gaseous nitrogen vapourized from commercial liquid nitrogen on the surface and piped underground to the fire zone. This technique is flexible and can be used to deliver at a wide range of flow rates.

Practical considerations for nitrogen inerting are discussed and operating procedures are recommended. Twenty-two case studies covering a range of geographical locations, mining methods and fire situations are included to illustrate recent international practice.

SOMMAIRE

L'inertisation est le procédé par lequel une quantité de gaz inerte est insufflée dans l'atmosphère en contact avec un feu afin de réduire l'apport d'oxygène et ainsi contrôler l'intensité du feu de manière à ce qu'il s'éteigne. L'inertisation ne peut être substitutée aux moyens habituels de lutte contre l'incendie, mais elle s'ajoute en tant que supplément de valeur aux nombreuses techniques pouvant être utilisées pour lutter contre l'incendie dans les mines souterraines. Les avantages de l'inertisation comprennent une réduction des dangers d'explosion, une dimunition de l'intensité des feux, une réduction des fuites d'air et une meilleure réhabilitation des quartiers incendiés.

Une partie du rapport consiste en une introduction générale sur les feux de mine, leurs causes, natures et effets. Les manières de lutter contre les feux et les précautions à prendre y sont revisées.

Le rapport décrit les méthodes actuellement employées pour lutter contre les incendies dans les mines, méthodes utilisant les techniques de l'inertisation telles que vues dans l'industrie Charbonnière du Canada. Ces techniques ont été employées dans beaucoup de pays pendant longtemps et elles sont maintenant bien établies dans les industries charbonnières les plus importantes du monde. Beaucoup de succès notables ont été accomplis et l'emploi de l'inertisation augmente dans les pays de l'Europe de l'Ouest aussi bien que dans les pays de l'Europe de l'Est.

S'appuyant sur des expériences passées ainsi que sur les conditions minières actuelles et présentement envisagées, plusieurs genres d'incendies peuvent survenir dans les mines souterraines Canadiennes. Ceux-ci comprennent des feux sur les chemins et à d'autres endroits accessibles sur les longues tailles, ou dans les arrière-tailles, les échauffements dans les parois, pilliers ou

autres endroits de charbon solide et dans les endroits déjà minés. Les techniques de l'inertisation appropriées aux conditions Canadiennes doivent donc avoir la capacité d'être utilisées pour éteindre les feux à découvert et pour prévenir ou contrôler les échauffements.

La disponibilité, le coût et d'autres facteurs ont amené l'industrie charbonnière à concentrer son attention sur trois procédés d'inertisation - produits combustibles gazeux, gaz carbonique et l'azote. Puisque des succès suffusants ont été accomplis avec ces procédés dans plusieurs parties du monde, on peut s'attendre avec confiance qu'un ou plus d'un de ces procédés sera appropriés aux conditions Canadiennes. Les mérites relatifs de ces trois procédés d'inertisation ainsi que leur système respectifs de rendement ont été revisés et classés selon que leur emploi a été jugé convenable aux mines souterraines de houille du Canada. La technique d'inertisation recommandée pour un emploi général dans l'industrie charbonnière du Canada est celle qui se caractérise par une évaporation à la surface d'azote livré à l'état liquide suivie d'une insufflation à l'état gazeux dans des conduites jusqu'aux points d'utilisation au fond. Cette technique est flexible et peut être utilisée à différents dosages de débits et de pression du flux d'azote.

Des considérations pratiques pour l'inertisation par l'azote sont discutées et des données d'opération sont recommendées. Afin d'illustrer les récentes pratiques internationales, on a inclus à ce rapport les études de vingt-deux cas couvrant différentes situations géographiques, méthodes minières et quartiers d'incendies.

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SECTION 1 - BACKGROUND

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SECTION 1 - BACKGROUND

1.1 INTRODUCTION

1.1.1 Objective

The objective of the study is to report to the Canadian coal mining industry the current status of mine fire control using inerting techniques. These techniques have been applied in many countries over a long period of time and are now well established in the major coal industries of the world. Many notable successes have been achieved and the use of inerting is increasing both in western and eastern bloc countries.

The report is designed to provide Canadian mine operators and rescue organizations with an account of experience gained principally offshore and to relate that experience to Canadian conditions.

Inerting is not a substitute for conventional fire control methods, but is a valuable addition to the range of techniques that can be applied to combat underground fires in coal mines.

1.1.2 Approach

The basic principles of mine fires and firefighting are reviewed, experience with the inerting is outlined and current inerting technology is described. The conditions prevailing in Canadian underground coal mines presently being worked and those anticipated to come on-stream in the next five to ten years are reviewed and fire control requirements for Canadian coal mines are identified. The inerting technique most suitable for application in Canada is recommended and operational procedures are described.

The study draws heavily on the experience gained in the fighting of mine fires in a wide range of mining conditions in several different countries. This experience is illustrated in a set of case histories that show the many different ways that inerting has been used to deal with various kinds of fire.

This report is intended to be used by operating mine managements and by the emergency services and only essential theoretical material is included. More detailed information on particular aspects of inerting can be found in specialist publications listed in Appendix A.

This report reviews inerting from a coal mining perspective, but the techniques are equally applicable to underground metal and industrial mineral mines and other work places where direct fire fighting is not always practical or feasible, for example, industrial locations where flammable gases, liquids and/or vapours may be encountered or where access for direct fire fighting may not be possible.

1.2 INERTING TECHNIQUES

1.2.1 Benefits of Inerting

Inerting is the introduction of an inert gas into an atmosphere in contact with a fire to reduce the level of oxygen and hence the intensity of the fire to the point of extinction. The benefits of inerting include:

a. Reduced Explosion Hazard

Depending on the composition of the atmosphere surrounding the fire and the levels of methane, hydrogen and carbon monoxide it is possible to determine the percentage of oxygen at which the risk of explosion is eliminated. Provided the concentration of oxygen is less than 4%, the atmosphere will normally be incombustible even in the presence of large percentages of hydrogen, methane and carbon monoxide. However, it must be remembered that the atmosphere in, for example, a sealed off area, is not always a homogenous mixture.

b. Reduced Fire Intensity

In a fire zone the initial combustion may well set alight materials and coal surrounding the fire. Inerting can reduce the extent of secondary combustion and can cool the area surrounding the fire zone after the fire itself has been extinguished by the reduction in the level of oxygen.

c. Reduced Air Leakage

Leakage of air into sealed areas in excess of the natural leakage may be reduced either by introducing inert gas into the leakage paths to replace air or by pressure balancing seals and providing a chamber of nitrogen on the outbye side of the seals. Both methods assist in the reduction of the intensity of combustion.

d. Facilitates Re-entry

Inerting facilitates re-entry into sealed areas by limiting the heating up of strata around the fire zone, minimizing the explosion risk and reducing the temperature. Inerting with gases does not damage exposed equipment and facilities, which can be put back into service without major refitting and avoids large post-incident costs such as would be experienced if water flooding were used.

1.2.2 Inerting Media

Historically three gases have been used in coal mine fire control:

- Gaseous combustion products (GCP)
- Carbon Dioxide (CO_2)
- Nitrogen (N_2)

A number of cases have been reported where the exhaust products from a furnace on the surface have been conveyed into a mine to combat fire. This method has disadvantages in terms of speed of response and control and is no longer normally applied. Current practice is to generate GCP in purpose built inert gas generators, usually fueled with kerosene, designed and constructed to be transportable into the mine and erected in a relatively short time. These machines are capable of producing large flow rates of a gas mixture low oxygen, consisting principally of nitrogen and water, with some carbon dioxide, carbon monoxide and other secondary products of combustion. The IGG enables GCP to be brought to bear on a fire in a very short time and for the flow of GCP to be controlled.

Carbon dioxide has been used in both solid and gaseous form to combat underground mine fires. Difficulty in access to the fire zone frequently precludes the use of solid carbon dioxide and generally carbon dioxide is evaporated and piped into the fire area in a gaseous state.

Nitrogen can be used for inerting mine fires either in liquid or gaseous form. The use of liquid nitrogen, transported into the mine in special tanker vehicles, is relatively limited. However, the use of gaseous nitrogen evaporated on the mine surface from commercial liquid nitrogen or in certain circumstances from liquid nitrogen separated from air on site, has been widely adopted for mine fire control in many countries. Evaporators are available in a number of designs and are now being purpose built for use in combatting underground mine fires.

The relative merits of these alternative inert atmospheres are discussed in Section 3 of the report.

1.2.3 Inerting Applications

The purposes for which inerting is used include the control of active fires and the prevention or control of heatings.

Inert gases can be applied to a fire in sealed or mined-out (gob) area or to an accessible fire. When dealing with an open fire it may be necessary to restrict the air flow to the fire so that a sufficiently high concentration of inert gas can be achieved around the fire. Where inert gases are used to prevent and/or control spontaneous combustion in a goaf area, they are generally introduced into the fire zone while normal ventilation is maintained in the airways.

Inerting techniques complement conventional firefighting practices such as direct attack with water or stopping-off with pressure balancing.

Although the principle of inerting is fundamentally simple, practical application in emergency circumstances can be complex. The technique should only be applied after the circumstances relating to a mine fire are understood. Any inerting application should be managed by engineers with a thorough understanding, not only of inerting technology, but of the principles of mine ventilation and the whole process of mine fire control. In Section 2 of this report some basic aspects of mine fire control are discussed to illustrate the many factors that have to be considered in responding to any fire incident.

In handling a major fire emergency, consideration should be given to obtaining specialized assistance at the earliest possible time. There are a number of organizations in North America and offshore that have staff with wide experience in this field, who can offer invaluable assistance. However, speed in responding to a fire incident is absolutely critical to success and local mine management must understand the theory and practice of mine ventilation and fire control and must make appropriate contingency plans for dealing with mine fire emergencies.

1.2.4 Status of Inerting

There are some differences between practice in Western and Eastern Europe. The principal coal mining countries of Western Europe - France, Germany and the United Kingdom - have adopted nitrogen as the preferred inert gas, whereas in the Eastern bloc, Poland, Czechoslovakia, and the USSR have made extensive use of GCP, though a trend toward the wider use of nitrogen in these countries is now emerging.

In Western Europe the manufacture and transport of liquid nitrogen to the minesite, its evaporation and delivery into the mine are now established technical and commercial practice and the major Western European coal industries have contractual arrangements with nitrogen suppliers to deliver liquid nitrogen and evaporators under emergency conditions.

Canada has a large nitrogen manufacturing capability and with its extensive oil and gas industry a well established nitrogen supply system and experienced nitrogen contractors. No difficulty is envisaged in establishing nitrogen supply arrangements in Canada comparable to those in Western Europe. In addition, carbon dioxide is obtainable on a commercial scale and inert gas generators could be made available if required.

Thus, the three principal inerting techniques have potential for application in Canada and are reviewed and compared in this report.

There have been many successes with inerting, though in some cases complete control of the fire has not been obtained, or a fire has reactivated when a sealed area has been opened up and/or inerting has ceased. Inerting is, however, a well established and important tool for mine fire control.

1.3 INERTING EXPERIENCE

Underground mine fire control by inerting has been practiced since the middle of the 19th century. Gaseous combustion products (GCP) are reported to have been used in 1849 and 1851 to control mine fires in Scotland, UK. The same technique is reported to have been employed in the USA in 1875 and in 1885 inerting was used to control a fire in Belgium. Inerting is said to have been applied in Russia in the early part of the 20th century.

Several examples of inerting with GCP and with carbon dioxide are reported during the first three decades of the 20th century.

Nitrogen appears to have been used first in Czechoslovakia where an improvised nitrogen generation plant was set up on a mine surface in 1949 to provide inert gas to combat an underground fire. The use of commercially produced nitrogen is reported from Czechoslovakia in 1955.

From the fifties to the present day, many cases of inerting have been reported using GCP, carbon dioxide and nitrogen. GCP have been widely used in the USSR and Poland as an inerting medium and on occasions in Czechoslovakia. Since the early 70's interest in the use of nitrogen has increased and there has been a decrease in emphasis on carbon dioxide. Germany has been prominent amongst the western coal industries in widening the application of nitrogen fire control methods. Apart from one trial with an inert gas generator, virtually all inerting in Germany in the 1970's and 80's has been with liquid nitrogen evaporated on the mine surface. Tables 1-1a, 1b and 1c summarizes the inerting incidents in Germany for the period 1974 to 1983.

The French coal industry employs nitrogen techniques on a regular basis and the UK coal industry has used nitrogen to combat several mine fires since 1980.

The eastern bloc coal industries continue to employ GCP produced by inert gas generators, though increasing interest in nitrogen appears to be developing in the USSR.

The majority of inerting applications in the past have been concerned with the control of mine fires in emergency situations. A relatively new development is the use of nitrogen as a means of preventing or controlling spontaneous combustion heatings. A number of successes have been reported with this technique in France, Germany and the UK.

Appendix A provides a review of international inerting experience up to 1974. More recent experience is illustrated in the Case Studies (Section 7) and in many technical papers (for selected references see Appendix E).

Inerting is well established as a means of fire control and materials and systems have been developed to a fully operational status and are commercially available in most coal mining countries.

TABLE 1-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

INERTING WITH NITROGEN IN GERMAN COAL MINES (1974 - 1983)

Incident #	Start Year	Start Month	Days Duration	N_2 Used (km^3)	Type of Fire*	Results	Remarks
1	1974	12	6	155	I	1 face saved	
2	1975	04	36	700	I	1 face saved	
3		09	22	550	I	1 face saved	
4		10	42	2 500	I	2 faces saved	1st inerting without seals
5	1976	01	3	310	I	explosion suppressed	
6		02	21	1 600	O	3 faces saved	1st use of F hose
7		07	3	120	O	1 face saved	
8		07	94	1 339	I	1 face saved	
9		10	28	545	I	1 face saved	
10		10	7	600	I & O	district sealed off	
11		12	14	3 700	O	district sealed, reopened 3 faces saved	1st use of 2F hose
12	1977	02	3	530	I	district sealed, reopened	air cooling
13		06	3	85	I	district saved	
14		06	5	100	I	1 face saved	air cooling
15		06	6	360	I	1 face sealed off	
16		08	45	1 052	I	1 face saved	fan failure
17		08	16	1 040	I	1 district saved	BTS evaporator used
18		08	4	345	O	1 face saved	
19		10	34	9 630	O	3 faces saved	
20		12	23	675	I	5 faces recovered	
21	1978	01	81	13 000	I	1 face saved	1st manless operation
22		03	7	829	I	1 face sealed, district saved	
23		05	338	10 880	I	1 face saved	
24		06	160	1 850	I	1 face saved	pressure balancing
25		08	80	6 800	I	1 face saved	
26		09	46	3 500	I	1 face saved	1st inerting through borehole
27		10	22	675	I	salvage, sealed off	
28		10	758	34 253	I	mining completed as planned	
29		10	13	767	I	salvage, sealed off	
30		11	51	3 270	I	1 face saved	
31	1979	01	157	7 000	I	face equipment saved	
32		01	2	148	I	stoppings advanced safely	
33		01	17	1 233	I	face equipment saved	
34		02	9	589	I & O	1 face saved	
35		03	6	300	I & O	explosion	hazard eliminated
36		07	2	9	I		
37		08	30	4 864	I & O		local inerting
38		09	449	6 465	I	maingate only saved	initial and preventive inerting
39		10	3	148	I	1 face saved	local inerting
40		10	22	871	I	explosion	hazard eliminated
41		12	10	507	I	1 face saved	local inerting
42	1980	02	1	16	I	district sealed off	
43		06	16	735	I	production maintained	preventive inerting
44		07	35	1 500	I	1 face saved	
45		08	82	6 422	I	production maintained	
46		11	117	2 951	I	production maintained	
47		11	48	1 902	I	production maintained	

* I = inaccessible
O = open

.....Cont'd

TABLE 1-1 (Cont'd)
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

INERTING WITH NITROGEN IN GERMAN COAL MINES (1974 - 1983)

Incident #	Start Year	Start Month	Days Duration	N_2 Used (km^3)	Type of Fire*	Results	Remarks
48	1981	03	2	47	O	gas ignition	
49		04	558	8 206	I	production maintained	preventive inerting
50		07	86	8 500	I	production maintained sealed off 09/81	N_2 flow 70-90 m^3/h had to be maintained
51		08	2	530	O	fire contained	
52		08	305	5 002	I	production maintained	recurrent inerting
53		10	74	3 549	I	robbing	
54		10	98	2 904	I	robbing	
55	1982	02	9	1 478	O	drift and machine saved	1st inerting of a full-face machine
56		02	189	3 324	I	production maintained	air evaporator (Linde)
57		03	19	2 132	O	drift sealed, adjacent heading continued	
58		07	30	1 271	I	face equipment salvaged	
59		11	253	18 634	I	1 face saved	room and target inerting explosion threat
60		11	45	1 141 +154**	I	1 face saved	target inerting N_2** from air separator unit GAN 800
61	1983	04	3	334	I	production maintained	target and roof inerting
62		06	212	9 089	I	production maintained	room and target inerting, seals and boreholes
63		11	35	2 749	I	production maintained	

* I = inaccessible
O = open

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UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

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SECTION 2 - MINE FIRES

2.1 GENERAL

Many thousands of mine workers have lost their lives during fires in coal mines. In addition, fires have frequently caused the abandonment of many mines or the sterilization of large reserves of coal. In one particularly serious disaster a fire in a coal mine in Manchuria during the Second World War resulted in the death of 1 500 miners. Thus the prevention and control of underground mine fires has both safety and economic implications.

2.2 CAUSES OF MINE FIRES

The main causes of underground mine fires include:

- Conveyors.
- Frictional ignition.
- Spontaneous combustion.
- Electrical equipment.
- Inadequate legal standards of enforcement that allow:
    - smoking
    - non-flameproof materials
    - unsuitable explosives
    - electrical equipment such as heaters and kettles.

2.3 TYPES OF FIRE

2.3.1 Small Fires

A small fire is confined to a localized and exposed surface. For example, a fluid coupling on a trunk conveyor may overheat and set fire to oils and fluids, pumps may run dry and stators or motors become ignited. Other examples include waste paper catching fire, and fires in small piles of coal or dust under conveyor belts, near loading points, or resulting from small falls.

Such fires should be tackled immediately on discovery by personnel at hand, using whatever materials are available. It must be recognised that a small fire is always potentially a large one.

2.3.2 Large Fires

A large fire is one which completely fills the roadway with flame, burning materials or fire gases and the floor, sides and roof of the roadway may be alight. Such a fire has the most serious implications on the safety of the mine because noxious fumes may spread through a considerable length of roadways, and the risk of explosions or 'flash overs' of fire gases is present. Thermal effects can influence normal mine ventilation, locally and throughout the mine.

2.3.3 Deep-seated Fires

Such fires (heatings) are located remotely from the roadways. Physical signs of fire such as flame and fire noise are absent, though other signs of fire may be present to some degree, such as smell, smoke, heat, condensation and the emmission of the products of combustion. Some of the signs such as heat and condensation may be absent in some cases, but others such as smell and the products of combustion, can be picked up in the mine ventilation, either from an identifiable spot - a seal or stopping - or, more usually, progressively along a section of roadway.

The location of a heating may be a few centimetres into the sides, roof or floor of the roadway or up to several kilometres from the mine workings themselves. There are many instances of heatings occurring in old workings or outcrops passing products of combustion considerable distances underground through old gob, broken ground, or old mine workings, to emerge in the active workings of a mine.

Most heatings consist of oxidization of strata (not necessarily of high carbonaceous content) causing a rise in temperature that drives off volatiles and partial combustion products. The rate at which the carbonaceous material is consumed is usually low. For example a heating producing a pollution of 100 ppm (0.01%) CO in a ventilation current of 9.4 m^3/sec consumes about 4.5 kg of fuel per hour.

2.3.4 Hanging Flames

This term describes the situation where methane has been ignited and flame has travelled into breaks in the strata, or gob areas, and continues to burn in places remote from the workings, or the original ignition point. Such a flame can burn indefinitely as long as methane and oxygen are available. The heat produced can weaken the strata and exposed carbonaceous materials can be set on fire. If the flame reaches any pockets of methane/air mixture, explosions can result. A hanging flame is unstable and can travel backwards and forwards along strata breaks. It is virtually impossible to extinguish such flames by direct means and the most effective treatment is by sealing off or inerting.

2.3.5 Gas Jet Fires

Gas flares are common in the oil industry when a well catches fire, but they also occur in mines when gas under pressure from a borehole drilled for shotfiring, methane drainage, or from strata breaks, is ignited. Should gas become ignited by shotfiring or open fire in a roadway where methane drainage is practiced, then a continuously burning gas flare can result. It is often possible to extinguish flares by direct means such as stopping the flow of gas by closing the stand pipe valve on a methane borehole, or by the use of dry powder extinguishers.

2.3.6 Concealed Fires

These are cases of 'open' fires which occur behind structures or lining of roadways and are not immediately detected. It is virtually impossible to extinguish such flames by direct means due to the problem of access.

2.3.7 Surface Fires

Smoke and products of combustion from fires on the surface can be drawn into the mine by the ventilation current. Traces of carbon monoxide may register on underground air sampling or monitoring systems. Building, grass or forest fires near a mine have also been known to pollute the underground mine ventilation system.

2.4 COMBUSTION PROCESS

2.4.1 Fire

Fire is essentially the reaction of substances with oxygen at a sufficiently high rate to produce increased temperatures. Many substances react with oxygen in the atmosphere, but at a rate insufficient to cause increase in temperature, for example, the rusting of iron. In the fire situation, the substance is called 'a fuel'.

For a fire to occur, the necessary elements are fuel, heat and oxygen. If any one is missing, then no fire can result. If a fire is burning and one of the factors is removed, then the fire will go out. The basis of fire-fighting is, therefore, to remove one or more contributing factors from the situation. The fire may be starved of air and hence oxygen, the fuel source may be removed or the burning materials cooled.

2.4.2 Fuels

Fuels encountered in underground coal mines can be divided into two categories, namely:

Indigenous (naturally present)

- methane
- coal
- oil (occasionally)

Imported materials (taken underground)

- timber
- plastic materials
- flammable liquids and greases
- others, eg paper

Imported materials such as timber supports, are usually the medium through which a small fire develops into a severe one. This problem can be designed out, to some degree, by limiting their use particularly at vulnerable places such as conveyor transfer points. All combustible materials taken underground should be fire resistant. For example, fire resistant fluids should be used for hydraulic power systems.

The indigenous fuel supply can be controlled to some extent. Methane hazards can be reduced by good standards of ventilation supplemented by methane drainage. Coal dust should be prevented from accumulating near any potential source of fire. The fire hazard of coal itself can be reduced by the construction of fire-proof linings at transfer and loading points.

2.4.3 Spontaneous Combustion

Certain seams are known to be prone to spontaneous combustion, whilst others rarely experience it. The most common cause is the leakage of air through carbonaceous strata. Good ventilation engineering and practices are the best prevention. Workings should be well ventilated with pressures across old workings or gobs etc. kept as low as possible. The siting and construction of air crossings and ventilation doors are important, and airways should be as free of obstruction as possible.

The second most common cause is the existence of crushed coal and other combustible materials in vulnerable places. Insufficient planning can give crushed coal conditions when small pillars of coal are left or when roadways meet at acute angles. Badly designed air crossings and other structures also cause coal to be crushed, allowing air to enter. Some heatings can be prevented by moving timber from waste areas and avoiding building loose coal into packs. Other causes include poor ventilation due to delay in sealing off old districts.

The presence of moisture in the coal or in the ventilation, and certain chemical substances such as pyrites appear to act as catalysts to induce heatings.

The oxidation of the coaly substance generates heat, and if this heat is not removed the oxidation process escalates and the temperature rises. This is the case where crushed coal is exposed to the seepage of low quantities of air.

The earliest sign of a heating is a characteristic oily smell in the ventilation. This is followed by a phenomena known as 'sweating' where condensation of water vapour is seen on the supports and linings of the adjacent airway. More physical signs develop as the heating progresses including an increase in temperature on the roadside near the heating, warm vapour emission points followed by wisps of vapour and, later, smoke issuing from the roadside. Finally, the temperature rises to the point of ignition and open fire breaks out in the affected area. These symptoms are summarized in Figure 2-1.

2.4.4 Products of Combustion

The gaseous products of a fire underground are:

- **Smoke**

  Smoke can vary from a light haze to a black mass through which it is impossible to see. Even though such smoke may be diluted as it travels outbye, it is possible for it to be clearly visible in all the airways up to the discharge from the main fan. Because of the associated thermal effect, it is common for smoke to form layers near the fire with dense smoke near the roof and fresh air near the floor of the airways. The smoke is likely to move backwards against the air current at the roof near the fire. Work in smoke usually involves the use of trained rescue teams equipped with smoke goggles and life lines.

- **Water Vapour**

  Water is formed during combustion. The inherent moisture in the strata is driven off by the heat of combustion, and water from fire fighting materials evaporates. High humidities are, therefore, usually experienced downwind of the fire. The atmosphere in static air in sealed off areas becomes fully saturated. This factor has a considerable influence on fire fighting operations and or rescue work following a fire. To a smaller extent, there can be condensation problems with equipment and materials.

- **Carbon Dioxide**

  Carbon dioxide is produced in large quantities by all forms of combustion. Readings of up to 5% are observed in the atmosphere downstream of the fire and may be in excess of 10% in fire sealed areas. At concentrations above 5%, carbon dioxide has the effect of increasing the breathing rate, and can be lethal at 10% and over.

- **Carbon Monoxide**

  Carbon monoxide, an inflammable gas, is formed by the incomplete combustion of carbon and oxygen. If the availability of oxygen is restricted as in the case of spontaneous combustion, the proportion of carbon monoxide increases.

Carbon monoxide is normally present in mine ventilation to the extent of several parts per million, and this 'norm' should be known for each mine or section. A significant rise above this 'norm' is one of the earliest signs of a fire or heating and each level of increase requires an appropriate response. A consistent reading above 50 ppm confirms the presence of active combustion and requires remedial action. For concentrations above 100 ppm standby cover by men with breathing apparatus is indicated and at concentrations over 200 ppm continuous work should be undertaken only by rescue teams with breathing apparatus.

Carbon monoxide is a highly toxic gas, and headaches and dizziness are experienced at concentrations above 150 ppm. Concentrations of 0.5% and over can cause unconsciousness within seconds and are fatal in a very short time. Details of the physiological effects of carbon monoxide are given in Figure 2-2.

- **Hydrogen**

Hydrogen is produced by fires when water comes into direct contact with hot carbonaceous material. It is produced when a jet of water hits burning coal, when grout is injected into heated strata, or when pools of water submerge burning materials. Usually the concentrations of hydrogen produced in the ventilation can be measured in parts per million, but in some circumstances, much higher quantities can be produced ranging up to 3% of the atmosphere.

Hydrogen is flammable at concentrations of 2% in air and becomes violently explosive at slightly higher concentrations.

- **Other Gases**

  The combustion reaction produces, in small quantities, a range of other organic gases including ethane, propane, ethelene and aromatics such as benzene and toluene. Should other materials particularly plastics, be involved in the fire, other toxic gases such as ammonia, oxides of nitrogen, hydrogen cyanides, phosgene, and hydrogen chloride can be produced.

The gas mixture produced by fires are very complex and requires both good facilities for analysis and an understanding of the hazards which are present. Many of the gases are flammable. In addition, methane may be present in the mine ventilation from other sources.

As an example the composition of the atmosphere at the time of completing a stopping during a spontaneous combustion incident was:

methane	0.95%
oxygen	10.7 %
carbon dioxide	8.5 %
carbon monoxide	1.41%
hydrogen	1.35%

This mixture approaches the explosive limit, and is lethal to personnel.

2.4.5 Spread of Fire

If not tackled promptly, a small open fire can spread downwind using whatever fuel source available. The heat developed will be intense enough to jump over non-inflammable sections of roadway. The rate of advance of the fire is related to the velocity of the ventilation, and the gradient of the roadway. Fire spreads typically at the rate of 30 m/h.

Some recent examples of the development of fires in mines include:

- An inspection at 03:00 hrs of the return roadway of a longwall face during salvage showed no fire. At 05:30 hrs, 5 m of the roadway were on fire and by 07:30 hrs 20 m were ablaze.
- A fire at an air crossing travelled inbye along the intake for an unknown distance producing intolerable smoke and heat for about 1000 m downwind.
- A small fire at the bottom of a return drift spread to envelope most of the drift and finally killed 16 rescue men when the products of combustion exploded during sealing off operations.
- A belt fire in a return drift about 1200 m from the shaft bottom, spread and produced temperatures which resulted in the death of 6 rescue men from the heat developed, as they were being lowered down a shaft.
- A fire in a control cabin near a shaft bottom, spread behind the roadway lining. Rescue teams took less than an hour to

extinguish the cabin fire, but it took a further six hours to put out the fire behind the roadway covering. The fire had spread about 60 m.

2.5 DETECTION OF FIRE

2.5.1 Human

Human beings are very effective fire detectors, using their facilities of sensing and behaviour in the following ways:

- **Smell**

  The human nose is a most sensitive detector, in some cases being more sensitive than the best scientific apparatus. It can detect some gases in concentrations as low as one part in ten million. Experienced mining people can detect an unusual smell, and suspect a fire, immediately upon entering a return shaft or airway, sometimes kilometres from the fire. The exact location of a heating, or an emission point of fire gases can be detected by 'sniffing'. The suspected area should be approached from the intake side since the sense of smell is deadened by prolonged exposure and an approach from the return side is less reliable. The ability to interpret smell into incident or fire types is gained only by experience.

- **Sight**

  Smoke and haze can be reliably identified by sight, and shaft important distinction between the two recognized. Glowing or flaming objects are readily sighted in dark surroundings.

- **Touch**

  In the early stages of fire or heating, there are points in the roadway which require attention and can readily be detected by touching the roof, sides or floor. The sense of detecting radiant heat can be developed as an aid to detecting remote hot spots.

- **Hearing**

  Hearing is important when smoke and heat prevents the other senses being used and the crackling of the fire, the spitting of the strata and the sound of roof falls (large or small) are valuable aids to safety and effectiveness.

The most valuable means of detecting a fire is a conscientious routine inspection procedure required by the safety rules of the mine. Far more fires have been detected by this means than by more sophisticated systems described later.

2.5.2 Canaries

Canaries, small birds or small animals have traditionally been used to safeguard men working in a CO hazard. The advantages of using canaries is that the indication is direct, continuous and positive. The disadvantage is that the bird may become acclimatized to CO and fail to respond to low percentages.

A bird will become distressed and fall off his perch after about 16 minutes at a concentration of 1,250 ppm whereas a man would not be in danger of collapse until after about 30 minutes at the same concentration. On the other hand, birds will not show any signs of distress until about the 500 ppm level is reached, whereas a man will start to experience headaches and dizziness at about 150 ppm.

2.5.3 Smoke Detectors

These detectors react to smoke particles rather than the presence of gaseous products of combustion. They are suspended from the roof at strategic points and are arranged to provide a local or remote alarm. Although designs have been improved in recent years, they have been found to be unreliable in use, responding to certain types of dust in the atmosphere and to shotfiring and diesel fumes.

2.5.4 Infra Red Detectors

The device is available in the form of an infra red camera which displays an image derived from the infra red light source on a small television screen. For rescue purposes they can be used to detect living casualties up to 40 m through dense smoke, but in the fire detection role they are used to examine the lining of roadways for any hot spots resulting from a heating. They can detect hot bearings and overheating electrical equipment. These devices have been used successfully but are expensive and require skill in operation. A simpler instrument is the infra red thermometer which will indicate the temperature of a remote object.

2.5.5 Air Sample Analysers

These detect fire and heatings from the presence of combustion gases, principally carbon monoxide. They can be either portable or fixed systems.

Portable CO meters give instantaneous readings and have either a low range (0 to 500 ppm) or a high range (0 to 2%). There are electronic instruments which give a digital readout, but can include other gases in addition to carbon monoxide.

Certain instruments have been designed to detect fire by responding to the combined product of combustion gases, including CO. These include instruments which give a more quantitative assessment of the intensity of the fire. In the course of development are multi-gas analysers which give the values of oxygen, CO and methane.

The electronic instruments can give remote alarm and indication, and can be incorporated into a comprehensive monitoring system. Tube bundle systems that draw samples to the surface from the mine through plastic tubes for analysis and indication, are installed in many mines and give a high standard of control. They suffer from the disadvantage of the time delay taken (up to two hours in some cases) for the sample to travel along the pipe to the analyser on the surface.

Monitoring systems require careful design to avoid false alarms caused, for example, by picking up shotfiring fumes. Their effectiveness depends upon having trained operators available to interpret the results obtained.

2.5.6 Carbon Monoxide / Oxygen Deficiency Ratio

The carbon monoxide/oxygen deficiency ratio is often used as an indication of the presence and intensity of fire, because it is independent of changes in ventilation. The ratio expressed as a fraction has the actual concentration of CO in the ventilating air as the numerator and the deficiency of oxygen compared with fresh air as the denominator, the whole being multiplied by 100 to give a convenient figure. Changes in the ventilation alters both the numerator and the denominator, hence the ratio is independent of ventilation changes, provided that the incoming air itself has no oxygen deficiency.

The interpretations of the $CO:O_2$ ratio into stages of a fire is shown in Figure 2-1, but since a plot of the ratio against time almost invariably follows the same pattern as the plot of the CO analysis, the additional information obtained from the $CO:O_2$ ratio is limited. Some authorities, however, believe it has value for appraising samples taken from methane boreholes or from deep wastes where the analysis is not representative of normal air.

2.6 EFFECTS OF FIRE

2.6.1 Fire Products

A fire underground can have a dramatic effect on all aspects of a mining operation. It poses a threat to the safety of men, leads to loss of production, restricts of movement and disrupts routine and often incurs a considerable financial expenditure.

The effects of mine fires are summarized in Figure 2-3.

- **Flame**

  Fire in a mine is a frightening experience and a section of roadway on fire throughout its perimeter is reminiscent of looking into a wood stove.

- **Smoke**

  The density of smoke depends on the intensity of the fire and the materials involved. For example, a fire of polyurethane foam or mineral oil generates very black smoke. Wood, on the other hand, usually produces a greyish smoke containing organic vapours which cause the eyes to run and the throat to become dry, usually well before there is any danger from toxicity.

- **Noxious Gases**

  The production of carbon dioxide, carbon monoxide and the chemical gases constitute one of the major safety hazards in a fire incident. It is for this reason that all personnel should be kept out of any part of the mine downwind of the incident. Access downwind should only be attempted by men wearing breathing apparatus. The atmosphere may still be lethal even though other signs of fire such as heat and smoke have been diluted or have dispersed. This hazard can exist in or at the top of an upcast shaft, in the fan drift and even on the surface of the mine downwind of the upcast shaft. Rescue teams should examine for toxicity; 'Danger Zone' notices should be posted at all doors or points of access into the return airway. Rigid instructions should be issued to all involved as to which parts of the mine are safe or unsafe. At one incident first reports said that the upcast shaft was polluted with 30 ppm of CO and therefore safe for men to travel. However, the first rescue team to descend took the precaution of taking a CO reading which registered 1700 ppm. Any person riding the shaft without breathing apparatus would have arrived at the pit bottom unconscious.

2.6.2 Explosion Risk

Open flame or incandescent materials constitute continuous sources of ignition. If inflammable gases, such as methane or the combustible products of combustion, are present in the right proportions, ignition will take place. If gas percentages are within certain levels, the ignition will be violent and can propagate into a coal dust explosion. Ventilation circuits may be damaged by the first explosion, and it is possible that there can be a succession of explosions. These are particularly dangerous to the rescuers dealing with the original incident.

To ascertain the explosion hazard at a fire, it is essential to have a reliable reading of the amounts of methane and oxygen present.

The properties of the atmosphere are most conveniently appraised by reference to the Coward diagram, shown in Figure 2-4. The diagram presumes that the atmosphere is composed exclusively of air and methane and that there may be some degree of oxygen deficiency present. The diagram is used for control purposes by plotting each successive gas analysis on a copy of the diagram and joining these points with a line. A trend will become apparent. If the line is slowly extending in Section C of the diagram it will eventually enter the explosive area. The rate at which the line is extending will also provide a forecast of when it is likely to happen. If the line is extending in part D then there is no danger of an explosion taking place.

If the atmosphere is diluted with fresh air through increased ventilation or reduced flammable gases then the trend of the plot line would be reversed towards the fresh air point, ie 21% of oxygen and nil % of methane. If this occurs when the plots are in Area A, C or D of the diagram, then the explosion hazard is being progressively removed. If, on the other hand, the sample analysis falls in Area B, then dilution with air can cause the atmosphere to move through the explosive range, a phenomenum which occasionally occurs in a mine fire situation, often with disasterous results.

If oxygen was progressively removed, then the line plot would move vertically downwards to the extinctive areas. This is what occurs when an inert gas is introduced to the fire situation.

2.6.3 Flammable Gases

In addition to methane, fire gases always contain other flammable gases, principally carbon monoxide and hydrogen which have different flammability characteristics to methane. For instance, hydrogen is explosive at concentrations between 2% and 70% when mixed with air. It is therefore necessary to consider the total amounts of flammable gases present when determining the explosion risk. The explosive limits of gas mixtures differ from those of individual gases.

Minicomputer programs are available which estimate the explosibility hazard and can be taken to the site of a major incident. When the computer system is not available, the Hughes and Raybold graphical method can be employed (Appendix B)

2.6.4 Fuel and Oxygen Rich Fires

A fire is said to be oxygen rich when the oxygen supply is more than adequate to react fully with the fuel decomposed by the heat from the fire. Conversely, a fire is fuel rich when the oxygen supply is insufficient to react fully with the available fuel. It has been found that the rate of spread of an oxygen rich fire is directly proportional to the velocity of the ventilation. The spread of the fire becomes very rapid and unaffected by ventilation speed in the fuel rich condition, and the admission of extra oxygen in these conditions can lead to explosions and 'flash overs'. Reduction in ventilation can cause a change from the oxygen rich to the fuel rich condition.

Referring to the Coward diagram (Figure 2-4), the area C represents oxygen rich condition and the area B represents the fuel rich condition. Any uncontrolled transition between one and the other produces a condition in area A, the explosive triangle.

2.6.5 Heat

The temperature of the burning materials in the mine depends largely on the quantity of the ventilation.

Air temperatures have been recorded at over 1000°C immediately downwind of a fire and at 200°C, 100 m downstream. Strata downwind will act as a heat sink but because of poor thermal conductivity will retain this heat. Higher strata temperatures in turn reduce heat transfer rates and extends the downwind elevated temperature zone. The high temperatures will destroy most materials, with even heavy steel structures distorting or disintegrating. Exposed strata will be cracked by the heat and will become unstable. For this reason, it is generally impossible to fight a fire from the downwind side.

2.6.6 Effects on Ventilation

A fire can affect the ventilation systems throughout the mine in two ways:

a. **Constriction** - this effect is brought about by a substantial increase in the volume of the ventilation as it passes through the fire. It expands due to the heat and the addition of fire gases. This increased volume in the roadway immediately downwind of the fire requires extra ventilation pressure to maintain the flow and consequently, the flow is reduced as it would be if resistance to the system was introduced. Since this is caused by the considerable increase in temperature, through the fire area, and since the temperature decreases some distance downwind of the fire, the constriction effect is therefore mainly generated in the vicinity of the fire itself, and the effect is always to oppose, and therefore to reduce the ventilation.

b. **Buoyancy** - this effect is due to hot air rising and being replaced by descending cold air. The buoyancy effect is associated with the gradient of the airways in the fire area. Its effect is minimal in level roadways and maximum in vertical shafts.

The buoyancy pressure generated by a severe timber fire in a roadway inclined at 1 in 2 would be about 0.5 kPa in the first 400 m downstream of the fire. The build-up of buoyancy pressure falls off downstream as the fumes cool. Buoyancy has the same effect as a booster fan installed in the ventilation circuit, blowing uphill. This means that an ascending ventilation current is boosted by the effect of a fire, whereas a descending air current is reduced or even reversed by the fire. It also follows that a fire at the bottom of an inclined roadway or shaft would have a far greater effect than a fire located at or near the top of the same shaft or roadway.

Examples of the buoyancy effect of fires are shown in Figures 2-5, 2-6 and 2-7. The situation shown in Figure 2-5 has arisen in a Nova Scotia mine.

The buoyancy effect can be very strong and can cause total disruption to the mine ventilation. In an extreme case (Black Heart Mine, Belgium) a fire in the downcast pit bottom reversed the ventilation throughout the mine and in the shafts themselves. Air was sucked backwards through the exhausting fan on the upcast pit top which was running at 1.5 kPa at the time.

In the event the main fan stopping during a fire incident, the mine ventilation system can be taken over almost completely by the buoyancy effect of the fire. There is a complete loss of control of the system and a highly dangerous situation can arise.

2.6.7 Smoke and Recirculation

All fires cause a considerable temperature gradient between the air at the roof of the roadway, and the air near the floor. This causes a layering effect of smoke and fire gases and it is quite possible for a roof of a roadway to be enveloped in dense smoke while there are clear air conditions at floor level. In level roadways this layering effect is prevalent for considerable distances downwind of the fire, but there is also a backing of the smoke layer against the ventilation current upwind of the fire. This layering is particularly persistent in inclined airways where the ventilation is downhill.

A development of the smokebacking phenomenon is the creation of two separate air currents in one airway, particularly if inclined. There can be a brisk current of fresh air going in one direction at floor level, and another brisk current of hot air, gases and smoke, going in the other direction at roof level with a clear line of demarcation between the two. This can be intensified if there are lines of steel ventilation ducts at roof level, or a clear passage for air above the roadway supports and linings. In an example at a recent fire in a slope drivage at 1 in 4, the downhill flow of fresh air was measured at 10 m^3/sec at floor level and a corresponding flow uphill of smoke and hot air at roof level.

If the fire intensifies still further, the re-circulation phenomenum will increase to include whole districts, airways or even the whole mine.

2.6.8 Falls

A major fire in a mine roadway will eventually burn away and weaken the support system. It will also dry out and fracture roadside strata.

This may lead to minor or major falls of ground. The following hazards arise:

- danger to personnel fighting the fire
- obstruction of access to the fire area
- obstruction and disruption to the ventilation system
- deposition of a mass of burning material at floor level
- production of blazing cavities at the roof and sides of the road.

This phenomenum is present in the fire area itself and in the area immediately behind the fire as it advances downwind. This means that fighting a fire from the intake side may have to be abandoned, as it becomes too dangerous for firefighters to work under an unsupported roof.

2.7 FIRE FIGHTING OPERATIONS

2.7.1 Principles

For combustion to continue a fire requires the continuous availability of fuel, oxygen and heat. When one or more of these factors is removed combustion will be stopped.

Fuel can be limited by starvation. Starving a fire means removing the available fuel, for example by draining fuel from a burning fuel tank. Alternatively, the fire may be moved away from combustible materials. This is attempted in the 'digging out' process which is occasionally used in mines (see 2.7.3 below).

Heat is limited by cooling. Cooling is a most effective way of extinguishing a fire, since combustion cannot continue when the materials which are burning are cooled below the flash point. This means that the fire must die away when the rate of heat production is exceeded by the rate at which heat is removed.

In this respect, water is a most effective coolant since large amounts of heat are absorbed in raising its temperature to boiling point and during the process of converting water into steam. It takes about six times as much heat to convert a given amount of water at its boiling point to steam as is required to raise its temperature up to boiling point. For this reason water is an effective fire fighting medium and should be applied to a fire in such a way that as much as possible is converted into steam rather than running away as free (but very hot) water. Steam formed in the early stages of a fire when the heat is considerable, will not be visible, but as the temperature of the fire falls, the steam will condense above the fire, a sign that it is being brought under control. However, if water is directed onto incandescent burning coal in a confined or unventilated space, the water reacts with carbon to form a mixture of hydrogen and carbon monoxide called water gas. Such a mixture of gases is highly inflammable and can be ignited by the burning materials, sometimes with explosive violence. A spray will cool the flames without reacting in the above manner. A concentrated water jet should not, therefore, be used in a confined space. It should be applied only on ventilated burning surfaces.

Oxygen is limited by smothering. The process of smothering a fire requires the exclusion of oxygen. This is usually achieved by stopping the flow of fresh air to the seat of the fire so that the combustion process or the emission of other gases, e.g. methane, reduces the oxygen content and the fire is extinguished. This is the objective in the process of stopping off or erecting fire seals and is the basis of the inerting process. The same effect can be achieved in small fires by smothering with dry sand or stone dust or by using foam type extinguishers.

2.7.2 Direct Means of Fighting Fires

All small fires and the early stages of large fires should be tackled by direct means. The action should be prompt and resolute, but with appropriate safety precautions, including, for example, isolating electric power supplies and leaving an escape route. Sufficient resources of men or materials are essential. Self-contained fire extinguishers are suitable for use on small fires.

Even a roadway which is well alight may be tackled directly by using water. In fighting a medium sized fire, the aim always should be to approach the fire from the upstream (intake) side. The return side of the fire is usually too hot to approach and it is only safe to approach by men wearing breathing apparatus. In order to mount an effective attack there must be sufficient water pressure in the fire fighting pipe range to operate several branches simultaneously.

There are a number of hazards encountered during direct fire fighting. There is the danger of the effects of heat and fire fighters should frequently change duties so as to limit the time of exposure. Coldness can be a problem because of strong air currents in the mine and waterproof clothing is, therefore, required. There is a smoke hazard that can be reduced by erecting brattice sheets to induce a strong ventilation flow at the roof of a roadway to prevent smoke layers forming or backing up. Variable spray nozzles have been found to be effective in forming a water curtain against smoke. An effective stream of cool fresh air can often be brought to the firefighters by laying ventilation ducting to a fresh air source.

If the fire fighting range, which should be a minimum of 64 mm in diameter, and preferably 100 mm to 150 mm diameter, passes through the fire area, the heat can change the water in the pipe to steam causing a very high pressure. Unless suitable precautions are taken, when the hydrant valve is opened, the flexible hose can be filled with high pressure superheated steam that can burst the hose and envelop the fire fighter in scalding steam.

As the fire develops, the most serious threat to the safety of the fire fighting team is the possibility of falling roof, often consisting of burning materials. It is this threat which usually limits attempts to deal with the situation by direct means and necessitates resort to indirect methods.

2.7.3 Digging Out

This is a term which means the removing of smouldering or burning material from a fire area, and is usually related to the damping down stages of open or spontaneous combustion fire which is thought to be very near the roadside. Digging out is not recommended because it is unpleasant and dangerous and the transport of the removed material is difficult. The process is usually ineffective since frequently the fire fighters are certain that the seat of the fire is within their reach, but as they advance they find the fire zone has receded. The dug out material can burst into flames over a large area. Digging out should only be attempted when the location of the seat of the fire has been clearly defined.

2.7.4 Indirect Means of Fighting Fire

Lining

Lining is often an effective way of stifling a deep seated heating and involves applying an airtight seal to the roof, sides and floor of the mine roadways in the vicinity of the incident. One method is to erect a lining of boards, blocks or metal sheets behind the supports of the roadway and to fill in the space between this lining and the strata with tightly packed sand or plaster. Where the road is supported by arched girders, the lining can conveniently be formed by placing close fitting wooden boards in the web of the arched girder over its circumference. In some cases a second line of supports can be erected inside the original support system to give a very smooth lined road when filled in by boards and backfilled. This is known as double lining or boxing.

When the sides and roof of the roadway are solid and stable, a spray lining is, perhaps, the quickest and most effective. Typical materials which can be used for this purpose are mixtures of cement or gypsum and fillers. Such materials are sprayed on using a special mixing and pumping machines with a short hose terminated by a nozzle. Such machines can spray about one tonne of dry powder per hour, mixed into a slurry form. Pneumatically operated machines perform the same function at a somewhat higher rate. It is difficult to achieve a truly airtight seal by the spraying technique if the roadside is heavily fractured or irregularly timbered. In all cases of lining, attention is required from time to time afterwards to ensure that the lining remains unbroken.

Lining should commence at least 10 m upstream of the incident spot and continue for a further 10 m beyond. This means that the minimum length is about 20 m though often large continuous lengths of roadways are treated, and in some cases the lining is continued as the working face advances.

In addition to the treatment of the sections of roadway where gob odours or smoke are apparent, lining is most effective in stretches of roadway where air is leaking into a fire.

Injection

Injection involves contacting all the breaks and cavities in the strata which are leading to and from the suspected fire area and filling these breaks with slurry type materials. This is achieved by systematically drilling into the strata. The first 1.3 m or so of each drillhole is drilled with a 60 mm (approximately) drill bit, and a 1 m standpipe is inserted. Grout is then injected to fill up any breaks contacted and to seal the standpipe. When the pipe is firm, the hole is continued, boring through the standpipe with a smaller bit. Each of the holes is drilled to about 4 m in length or until the major breaks have been contacted. The injection of slurry is then repeated until there is no further flow of slurry into the strata.

A rectangular pattern should be attempted, e.g. 2 holes on either side of the road and 2 into the roof; not forgetting 2 further holes into the floor to check for floor breaks. This drilling pattern should be repeated at about 5 m intervals over the affected or suspect lengths of roadway. This ensures that a good length of consolidated ground is formed. Again, a minimum of 20 m applies as the need to treat both intake and return accesses to the fire area.

A variety of materials can be used in the preparation of the slurry, whose principal property should be the ability to act as a permanent solid filling in any breaks but on the other hand, the slurry should be able to move considerable distances through small breaks, so that a viscous property is also required. Materials commonly used are mixtures of cement and stone dust or other proprietary materials based on cement or fly ash. Gypsum plaster is not such a suitable material for this purpose, since it does not travel far before setting, and also disintegrates into dust when subject to the high temperatures which may be encountered. On the other hand, bentonite (Fullers Earth) has the ability to travel long distances in the strata, but tends to slump away over a length of time. An extensive range of grout pumps are available having capacities ranging from 2200 L/h to 8000 L/h at pressures ranging from 1100 kPa to 5000 kPa. The minimum installation for injection usually consists of a mixing tank feeding a high pressure pump.

Inerting

This method can be classed as an indirect means of fire fighting. It is fully discussed in this report.

Stopping Off

Stopping off means plugging all roadway accesses leading to and from the fire area with dams, seals, or stoppings so that it becomes impossible for any air to reach the fire, or any of the products of combustion to be emitted into the ventilation. The fire is thus smothered by depriving it of oxygen.

The advantages of stopping off are primarily that of safety as stoppings are usually erected at least 150 metres away and in some cases can be as far as 1 km to 3 kmfrom the fire.

It is also efficient in terminating what may have become a serious or prolonged fire situation which has not responded to other methods of treatment. Stopping off is often a permanent cure and the chances of further serious trouble are small. Stopping off can be cheaper than prolonged treatment by labour intensive methods.

However, there are serious disadvantages including the almost certain loss of large amounts of capital equipment in the sealed area, the loss of working reserves of mineral and the sterilization of development areas. The control of a fire sealed area is also difficult in broken ground, and the recovery of a district once sealed can be difficult and may be unsuccessful if a fire, particularly spontaneous combustion, has been involved. The establishment of a large sealed area in a mine brings with it other implications for ventilation and water control.

A stopping should be airtight and this can be achieved by the construction of a brick or block wall, or even a wooden structure. Where any signs of fire has been, or may be present, then an explosion proof stopping is required. These can be massive constructions of sand or other materials about 8 m to 10 m long, or more usually walls about 4 m to 5 m apart filled with cement or gypsum plaster materials. The length of the stopping should be slightly greater than the average of the width and height of the roadway in which it is erected. Since the plaster and block materials are not always available, many stoppings are placed with mine dirt obtained locally and placed between packed walls. This type of stopping is neither airtight nor explosion proof.

The proper selection of the stopping sites is most important and should be undertaken both from a careful study of the full working plans of the mine, to ensure that every access to the fire is properly sealed and to ensure that the stoppings will not be bypassed by unconsolidated ground or old roads etc. and from a careful examination of the selected sites underground by an experienced person to make sure that the strata is solid where the stopping is to be constructed, and that heavy structures such as conveyor drives are avoided.

The walls of a stopping can be made of any materials which will contain the plaster fill until it sets solid. Sand bags, bricks, blocks, wood or metal shuttering have all been successfully used. The inclusion of access tubes is most important as they permit a continuing flow of ventilation round the district whilst the stopping is being completed, thereby virtually eliminating the risk of explosion, and permitting examination of the inbye wall whilst the stopping is being constructed. Typically access tubes varying from 500 mm diameter to 2.2 m x 2.2 m are used, often incorporating a massive explosion proof inbye door to facilitate rapid closure and the possibility of future re-entry into the sealed area. Air sampling tubes and water or gas drainage pipes are usually left through the stoppings.

The plaster filling material is either of the gypsum or cement based type. Bulk filling additives such as fly ash or sand are often included. The plaster mix is supplied from special placer machines which consist essentially of a metered mixing compartment feeding to a high pressure pump. According to the type of machine used and the type of plaster material, the placer can be situated from 20 m to 1000 m from the stopping site, 50 mm to 64 m diameter hose being commonly used to transport the plaster mix which has a variable setting time from 10 minutes to over one hour.

The key to the construction of the stopping is the preparation of the walls and the installation of fittings. The walls should be built on a firm foundation and should be made leak proof on each side and especially into the roof. This may entail removing covering sheets and timber and loose strata. Whilst this work is in progress, sufficient quantities of plaster powder should be brought to the placer position to complete the stoppings, allowing for loss due to leakage. This work often takes about one day to complete. Once pumping starts, it should be continuous as far as possible as interruptions tend to lead to the setting of the plaster in the machines and pipes and other mechanical troubles. Should there be an interruption, the machine pipes should be immediately flushed out with fresh water. It is important that the stopping is topped up with plaster right up to the roof of the roadway. This can be a difficult operation and it usually takes several hours to complete the last 0.5 m of the stopping, due to leakage at roof level. It is often necessary to pump through a probe pipe which delivers plaster to the highest point in the stopping. The final job is to seal the access tubes in all stoppings at the same time. Following the final sealing, it is usual for the mine to be abandoned for up to 24 hours to avoid the risk to personnel whilst the sealed atmosphere passes through the explosive range.

After the mine is re-opened, a continuing control of the stoppings is required to ensure that they remain secure and airtight and that air samples are taken at regular intervals from behind all stoppings, and in the local airways to monitor the fire activity which has been sealed off and any possible pollution of airways as a result of leakage from the sealed area.

2.7.5 Practical Aspects

Smoke control is a prerequisite for any fire fighting operation and suitable arrangements must be made when fire fighting activities are being prepared.

If a fire fighting range passes through a fire area water may be turned to steam which could be released under high pressure onto the fire fighters. Special mechanical contrivances, or the controlled opening of hydrant valves can minimize this hazard.

Fire fighters may become soaked with water and be badly affected by coldness on the way out of the mine. For this reason a supply of waterproof gear, dry overalls and dry blankets, should be provided for fire fighting teams.

Because of their special training, rescue men may be used as fire fighters both in fresh and polluted air. Except in life saving situation, standard rescue procedures must be observed to ensure that the rescue teams work in safety. There are limits to the capabilities of rescue men. Rescue procedures require that the operational times of rescue teams becomes increasingly limited with increasing temperature and this fact should be included in any rescue team roster which is prepared.

2.7.6 Pressure Balancing

Where a fire is controlled by sealing off, the reduction of air leakage into the sealed area is critical. If leakage is persistently troublesome, then pressure balancing may be applied. The technique involves the construction of an airtight wall at least 5 m in front of the main stopping, and a positive or negative air pressure is artificially induced inside it by means of a duct system. The duct leads to a remote fan, or by taps into a suitable mine ventilation pressure elsewhere. The pressure across the main stopping from the sealed area to the artificially formed chamber is brought to zero. Leakage across the main stopping will cease. Techniques are available by which the pressure in the pressure chamber can be automatically adjusted to ensure consistently stable and static conditions.

2.8 FIRE PRECAUTIONS

2.8.1 Introduction

Many planning and operational factors contribute to minimize fire and related major hazards at a mine. Several of the factors are discussed below, not necessarily in order of priority.

2.8.2 Mine Planning

Many mining hazards can be eliminated during the planning of the mine layout and its working methods. A careful study of a proposed development plan can reveal points of weakness that could be conducive to or aggravate an incident. This particularly applies to the prevention of spontaneous combustion incidents.

2.8.3 Fire Fighting Arrangements

All parts of a mine must have efficient fire fighting installations. On the surface, sufficient water supplies from a reservoir or from pumped sources must be available. Fire fighting ranges must be of adequate size, preferably 100 mm and 150 mm diameter with strategically placed hydrants, hoses and control valves. All intake airways and as much as practicable of the return airways in the workings districts should be equipped with fire fighting ranges.

2.8.4 Education and Training

Mock emergencies are a most valuable method of teaching and checking emergency arrangements. There should be instructions on the alternative means of egress from the mine and the working of the Mine Emergency Scheme. Periodic discussions and briefings should be held with supervisors and hourly paid employees.

2.8.5 Ventilation Contingency Plans

When a sudden emergency arises, it is virtually impossible to assess the implications to the mine as a whole. Preparations for these events can be made by preparing contingency plans for incidents in various locations on the ventilation system and for other emergency situations. This work can be assisted by computer exercises that will show the most dangerous situations and the potential remedial steps.

2.8.6 Stone Dust Barriers

The value of stone dust barriers cannot be over emphasized in limiting the spread of mine explosions. The installation of stone dust barriers should be liberal, their conditions should be regularly checked, and they should be moved forward as the workings develop. Additional barriers should also be erected as a safeguard during the course of an incident. Water trigger barriers perform the same function but are still undergoing technical development.

2.8.7 Fire Proof Zones

Fire proof stretches of roadways may be constructed with the aim of containing any major fire to certain sections of the mine or roadway. Because of the thermal transmission by the ventilation, such zones typically have to be 100 m to 200 m in length and must be completely free of combustible materials including flammable conveyor belting. This technique justifies further research.

2.8.8 Isolation Doors

The installation of explosion proof and fire proof isolation doors which can be closed following a severe incident, serve a similar purpose to the fireproof zone. They also restrict the products of combustion from polluting other ventilation circuits and so assist access and escape. Such installations are expensive and require careful design so that obstacles such as belt structure or rails can be quickly dismantled allowing the doors to be shut in an emergency.

2.8.9 Mine Emergency Scheme

Development of a Mine Emergency Scheme is a prerequisite to any disaster planning. The Scheme should be designed in detail to provide for safe withdrawal of men from the affected part, immediate attempts to treat the incident on the spot, notification to the surface, and progressive build-up of resources to deal with the incident.

2.8.10 Mobile Laboratory Facilities

A scientific back-up service is essential to the control of an incident, particularly in relation to the rapid and accurate analysis of air samples, and the interpretation of the analytical results. It is not always possible to have a sophisticated scientific control at each mine, but a mobile system should be available and kept in a state of constant readiness to be taken to any incident. This can take the form of a mobile laboratory or of transportable units which can be taken by truck or plane and installed at a suitable location in a mine. The mine should prepare for the arrival of the units by having services of water, power and telephone previously supplied to the selected laboratory site or office.

2.8.11 Prepared Stopping Sites

Stopping sites should be pre-selected so that they could contain any major incidents. It is preferable that primary sites are selected inbye, together with secondary sites further outbye, to contain a larger incident, or as a line of retreat should this be necessary. Sites with good access for men and materials should be selected, since the return airways may not be available for men or material transport because of smoke or pollution. In some cases these stopping sites can be partially prepared to ensure that stopping off can be rapid if their use is called for.

2.8.12 Self Rescuers

Self rescuers carried by mine workers have already been responsible for saving many hundreds of lives underground and their universal use is recommended. Simple filter type rescuers which remove carbon monoxide serve most purposes, but rescuers and escape sets which contain a supply of oxygen are even more useful but tend to be heavy, expensive and more difficult to maintain. The issue of self rescuers necessitates a good standard of initial training and regular refresher courses.

2.8.13 Mines Rescue Service

The Mine rescue services and the public emergency services need to be involved in any contingency planning for a major event. Not only will they be able to give good advice and information concerning the most effective way in which their services can be used, but they need to be informed of the various accesses into all parts of the mine. Because of smoke or damage, they may have to gain access to the affected area from shafts or drifts which are many kilometres away, and the rescue organization needs to be prepared for such an event.

2.8.14 Ventilation Monitoring

In theory, comprehensive monitoring systems will give immediate display and print-out information on the state of the ventilation and the level of impurities at that particular moment of time for all significant parts of the mine. If this was so, then such a system would be invaluable in giving warnings of impending events or give an accurate picture of the effects of a fire incident, and thereafter show the progress or otherwise of the containment of the incident.

In practice there are many occasions when this does not happen, and the information can be two or more hours out of date, incomplete, or erroneous due to pollution or dilution of the samples analysed or faulty hardware and software. Such a situation can lead to misreading of the situation with possibly dangerous results. Extreme care must, therefore, be taken in accepting or interpreting information from monitoring systems and the reading should be backed by routine analysis of air samples taken by hand or from the readings of reliable on-the-spot instruments.

Nevertheless, once a monitoring system has been 'tuned' to an incident, then the benefits in safety and efficiency are considerable.

2.8.15 Communications

Communications problems are always the most difficult at every major incident. The problem hits hard at all levels, ranging from the human aspect of being able to contact the people you want to, when you want to, and ensuring that the information and instructions passing both ways are properly understood. On the technical side, there is always a problem of having sufficient numbers of telephone lines and radio channels available to deal with the vast amount of traffic passing between specific points. Very often the communication system gets completely blocked and breaks down. Finally, there is a need to have the correct sort of communication system for the type of incident, e.g. telephone and teleprinter facilities, mobile radios and pagers for travelling personnel.

Communication problems can be designed out in the planning stages to a large extent by allowing spare capacity to greatly increase point-to-point contact if needed. Emergency communication equipment should be held by rescue services to supplement heavily loaded systems during emergencies.

2.8.16 Incident Management

Efficient management control is fundamental to the effective treatment of major or minor incidents. The levels of responsibility and lines of command should be establishing beforehand. A system of overall control must be arranged for incidents which affect several mines and the structure should be able to accept the additional and unfamiliar services and people who become involved at an incident.

The management structure included in any emergency scheme must take into account that certain management staff may not be available at the time of the incident.

A fundamental principle of any emergency scheme is that at all times an identifiable controller is in charge of the incident. In performing his duties he should be assisted by an incident control committee representing all interests and expertise relevant to the particular stage of an incident.

Apart from the initial stages of an incident when continuous attention of all those who are available and competent to help is required, the control of the incident should be agreed at specially called policy meetings at appropriate regular intervals, and in the intervening periods the controller with a small team of operatives and advisors should implement the agreed plan.

2.9 REFERENCE MATERIAL

Further information on many of the topics addressed in this section can be found in the following publications.

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Blackwell, J.R. and Roberts, A.F.
1969 **The Possibility of the Occurrence of Fuel Rich Mine Fires**; Min. Eng. (UK) September

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1969 **The Automatic Control of Pressure Balancing Chambers**; Min.Eng. (UK).

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1962 **Memorandum on Sealing Off Fires Underground; Inst.** Min.Eng. (UK).

Thurton, F.B.
1981 **Colliery Explosions and Fires. Their Influence upon Legislation and Mining Practice;** Min. Eng. (UK).

FIGURE 2-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

DEVELOPMENT OF SPONTANEOUS COMBUSTION

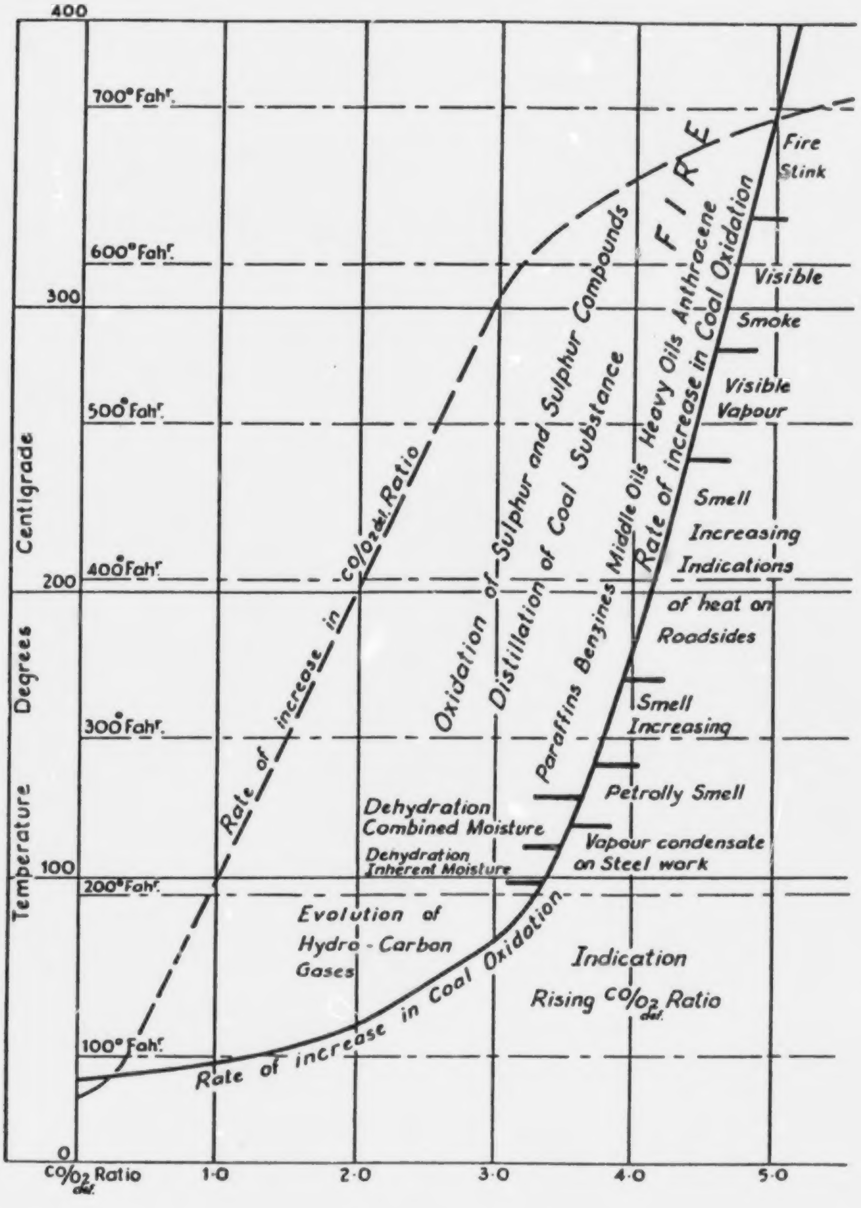
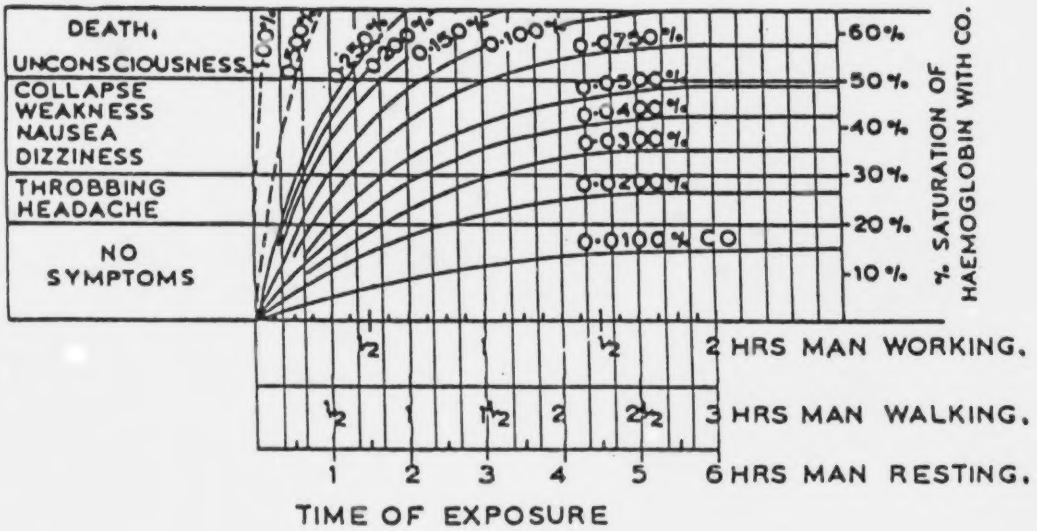


FIGURE 2-2

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

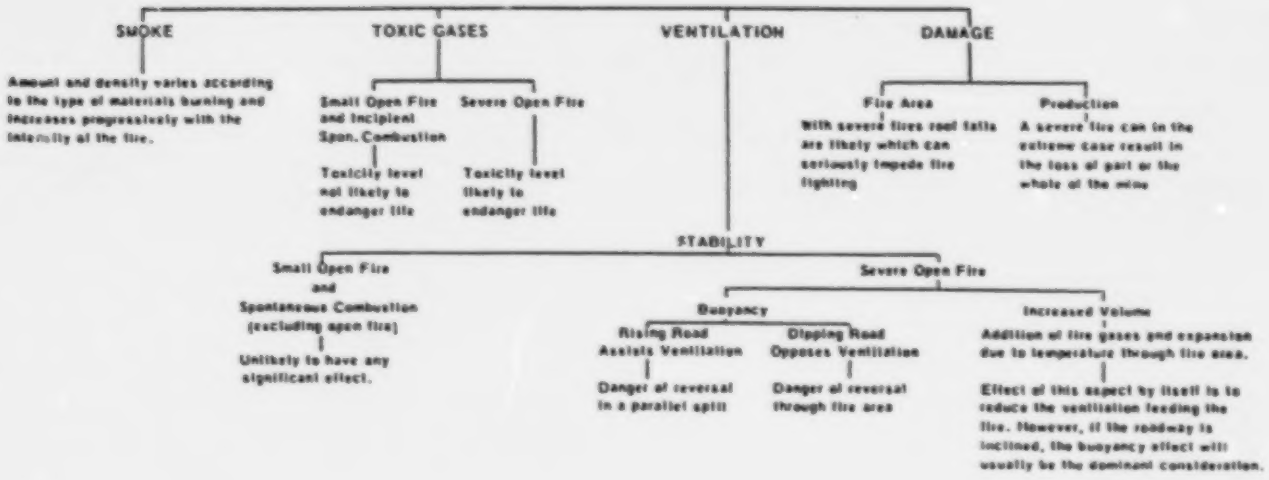
EFFECT ON MAN OF VARIOUS CONCENTRATIONS OF CARBON MONOXIDE



NOTE:- 0·0001% = 1 PART PER MILLION (P.P.M.)

FIGURE 2-3
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

EFFECTS OF MINE FIRES


SMOKE
TOXIC GASES
VENTILATION
DAMAGE
Small Open Fire and Incipient Spon. Combustion
Severe Open Fire
Fire Area
Production
STABILITY
Small Open Fire and Spontaneous Combustion (excluding open fire)
Severe Open Fire
Buoyancy
Increased Volume
Rising Road Assists Ventilation
Dipping Road Opposes Ventilation

Amount and density varies according to the type of materials burning and increases progressively with the intensity of the fire.

Toxicity level not likely to endanger life

Toxicity level likely to endanger life

With severe fires roof falls are likely which can seriously impede fire fighting

A severe fire can in the extreme case result in the loss of part or the whole of the mine

Unlikely to have any significant effect.

Danger of reversal in a parallel split

Danger of reversal through fire area

Addition of fire gases and expansion due to temperature through fire area.

Effect of this aspect by itself is to reduce the ventilation feeding the fire. However, if the roadway is inclined, the buoyancy effect will usually be the dominant consideration.

FIGURE 2-4

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

COMPOSITION AND EXPLOSIBILITY OF MIXTURES OF METHANE, OXYGEN AND NITROGEN

(Inflammability triangle - after Coward)

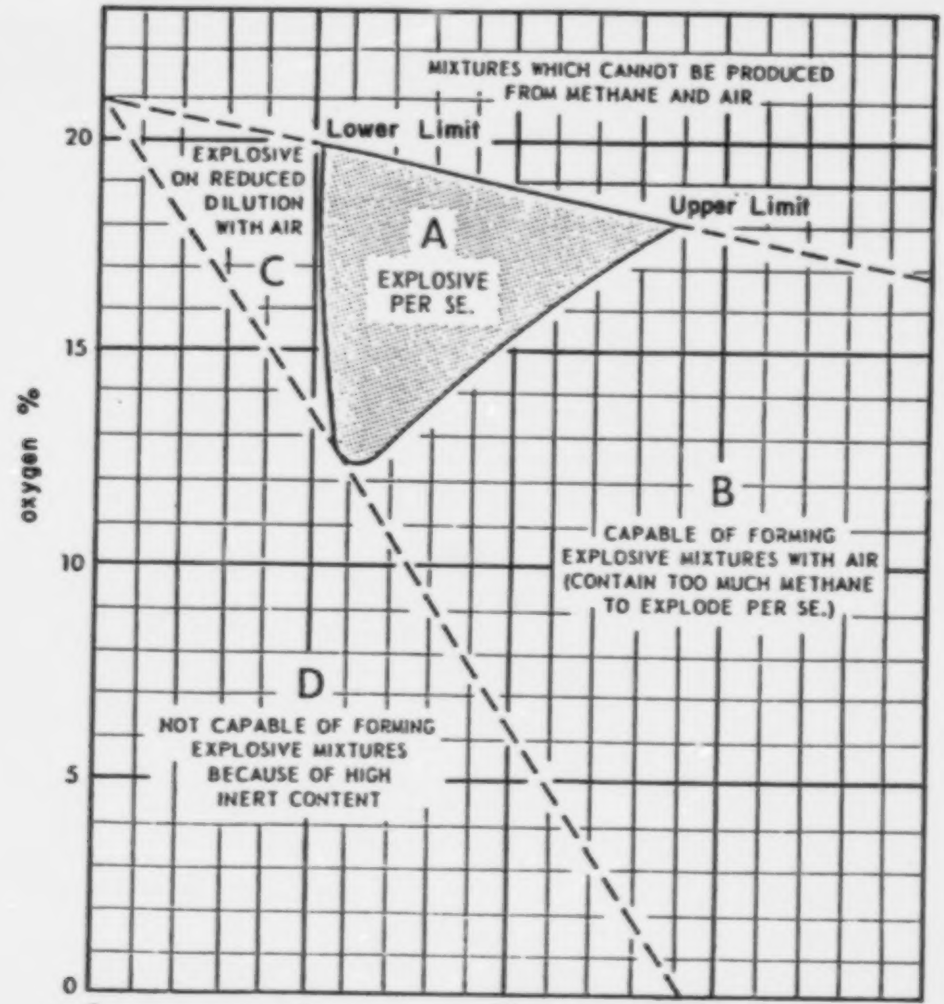
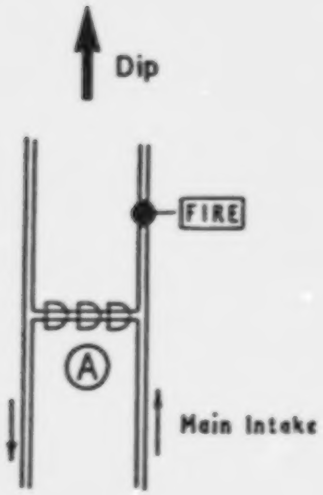


FIGURE 2-5
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

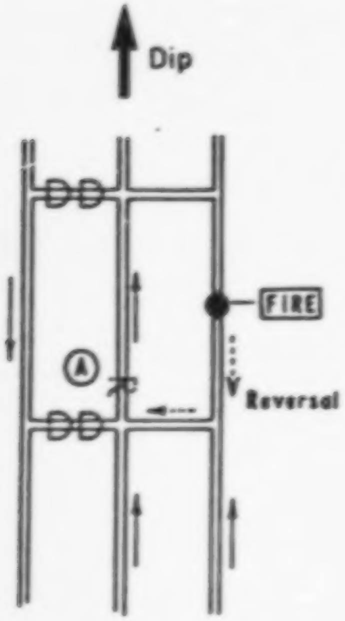
SEVERE FIRE IN INCLINED 2-ENTRY ROADWAYS



Note: Risk of reversal if,
a) Doors 'A' sealed
b) Main Fan stopped

FIGURE 2-6
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

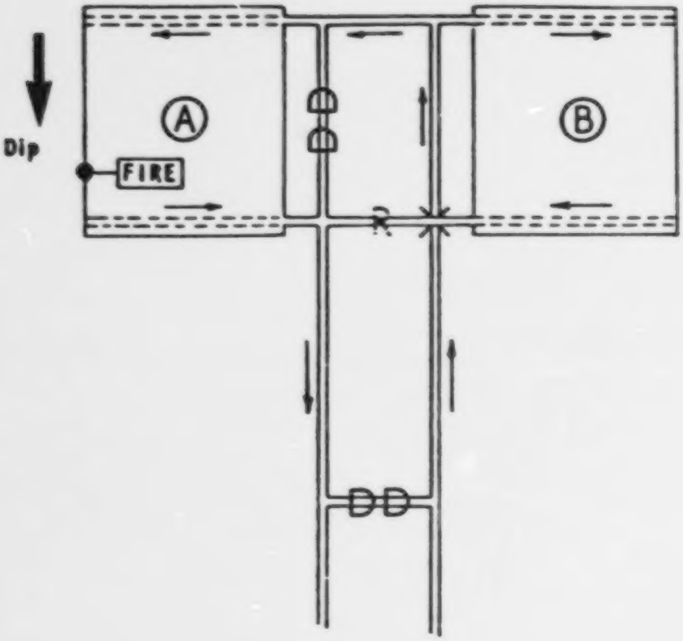
SEVERE FIRE IN DIPPING 3-ENTRY ROADWAY



Note: Danger of reversal through fire area resulting in
a) Increasing ventilation in adjacent intake
b) Some recirculation of the fire gases
Action Install regulator at point 'A' to maintain stability of vent.

FIGURE 2-7
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FIRE ON DESCENSIONALLY VENTILATED LONGWALL FACE



Note: Fire will reduce and possibly reverse ventilation in District Ⓐ and increase ventilation in District Ⓑ Regulators necessary in Ⓑ to stabilise ventilation

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UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SECTION 3 - CANADIAN UNDERGROUND COAL MINES

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(Following Text)

3-1 Canadian Underground Coal Mines

SECTION 3 - CANADIAN UNDERGROUND COAL MINES

3.1 GENERAL

There are active underground coal mines in three Canadian provinces: Alberta, British Columbia and Nova Scotia. The scale ranges in size from very small operations to major undersea mines. Mining methods include room-and-pillar, hydraulic and advance and retreat longwall.

In Canada, mine fires have been initiated by spontaneous combustion, conveyors, electricity and frictional ignition. Well developed mine fire fighting and rescue services exist in each mining area, but geographical separation between these areas limits collaboration in dealing with emergencies. Techniques employed to date to fight major fires include the use of water, foam, stopping off and in one case inerting with carbon dioxide. There is no record of the use of nitrogen or GCP for fire control.

In this section, current Canadian underground coal mining activities and proposed mine developments that may commence in the next 5 to 10 years are reviewed since it is against this background that the suitability of inerting techniques must be judged. A survey of active and proposed mines in provided in Table 3-1.

3..2 ALBERTA

3.2.1 Active Mines

Underground coal mining operations are conducted at the Smoky River Coal Limited property at Grande Cache.

A drift mine #9B, is working the No. 4 Seam by room-and-pillar methods and annual production is approximately 1 Mt of raw coal. Mining equipment includes continuous miners, roof bolters and shuttlecars. Development is by three and five entry roadways and pillars are recovered on the retreat. Material transportation is by free steered diesel and battery vehicles.

Pipe ranges of 50 mm or 75 mm diameter are installed to each active working place.

Some minor fires have occured and in one major incident, in 1980, 2R-4 Mine had to be abandoned when a fire could not be brought under control by direct attack with water and foam or by subsequent attempts at sealing off.

3.2.2 Proposed Mines

Future activities in the mountains/foothills region may include operations at SRCL's #9G Mine and Luscar's Coal Valley Mine, employing technology similar to that currently in use at SRCL or longwall methods. In addition, Cardinal River Coal have commenced development of an hydraulic mine near Hinton.

3.3 BRITISH COLUMBIA

3.3.1 Active Mines

Underground coal mining takes place at Sparwood where Westar Mining Ltd. operates two underground mines: Balmer North is a room-and-pillar mine and Balmer South is an hydraulic operation, both working #10 Seam.

Balmer North

This drift mine is accessed by rock tunnels and works room-and-pillar methods. Annual production is approximately 250 000 t of raw coal. Mining equipment includes continuous miners and shuttlecars. Development is by two or three entry roadways and floor coal is removed on the retreat. Material transportation is by free steered vehicles. Pipe ranges of 50 mm and 75 mm diameter are installed to each active working area. There has been no reported spontaneous combustion at this mine.

Balmer South Hydraulic Mine

This is an hydraulic mine accessed by rock tunnels and in-seam entries working the #10 Seam by hydraulic methods. Annual production is approximately 1 Mt of raw coal. Mining equipment includes monitors, breaker feeders and flume coal transport with continuous miners for development. Sub levels are driven at 30 m - 40 m intervals and the intervening coal blocks are recovered on the retreat. Material transportation is by free steered vehicles. Pipe ranges of 150 mm and 200 mm in diameter are installed to each active working place. Total ventilation flow is about 100 m^3/s. The most distant projected working place will be about 3 km from the surface.

A number of spontaneous combustion induced fires have occurred at this mine. Under prevailing mining conditions, the coal appears to be liable to spontaneous combustion. Sections of the mine have had to be abandoned following fire incidents. As a precautionary measure pillars are integrated into the mine design to form barriers between mining blocks.

3.3.2 Proposed Mines

Projects currently under consideration by Westar Mining include a multi-slice longwall or hydraulic mine at Baldy Ridge. Further west in the province, Norco, David Minerals and Wolf Mountain Coal are considering or developing underground mining properties. The technology to be employed will probably include room-and-pillar and longwall methods.

3.4 NOVA SCOTIA

3.4.1 Active Mines

The principal mining activities are on Cape Breton Island where Cape Breton Development Corporation operates three underground mines in the Sydney/Glace Bay area and a small mine is operated by Evans Coal Mines Ltd. in Inverness County.

CBDC's underground operations comprise three under-sea mines:

Prince Mine

This is an under-sea drift mine accessed by slopes, working the Hub Seam by short longwall retreat faces. Annual production is approximately 0.7 Mt saleable. The faces are equipped with double ended shearers, armoured flexible conveyors and powered supports. Material transportation is by rope haulage. Total mine ventilation is 76 m^3/s. Pipe ranges are provided into each face and development with diameters of 50 mm and 100 mm. The distance from the surface to the projected most in-bye working point is approximately 7 km.

No. 26 Mine

This is an under-sea mine accessed by shafts, working the Harbour Seam by longwall advancing faces. Annual production is approximately 0.7 Mt saleable. Faces are equipped with double ended shearers, armoured flexible conveyors and powered supports. Material transportation is by trolley locomotive, diesel locomotive and rope haulage. Total mine ventilation is 188 m^3/s. Pipe ranges are provided into each face and development with diameters of 50 mm and 100 mm. The distance from the surface to the projected most in-bye point is approximately 12 km.

Lingan Mine

This is an under-sea drift mine, accessed by five slopes, working the Harbour Seam by longwall advancing faces. Annual production is approximately 1.2 Mt saleable. The faces are equipped with double ended shearers, armoured flexible conveyors and powered supports. Material transportation is by rope haulage and diesel locomotives. Total mine ventilation is 205 m^3/s. Pipe ranges are provided into each face and development with diameters of 50 mm and 100 mm. The distance from the surface to the projected most in-bye working point is approximately 6 km.

Evans Mine

This is a small drift mine accessed by slopes working by room-and-pillar with scrapers. Annual production is approximately 50 000 t The distance from the surface to the most distant working place is about 1.5 km.

3.4.2 Proposed Mines

Major projects currently under consideration by CBDC include three new mines: Lingan-Phalen, Glace Bay and Donkin-Morien.

Lingan-Phalen

This will be an under-sea drift mine to be accessed by four slopes to work the Phalen Seam by longwall retreat faces. Annual production is projected at 1.5 Mt. Faces will be equipped with double ended shearers, armoured flexible conveyors and powered supports. The distance from the surface to the projected extent of the workings is about 8 km.

Glace Bay

This is a proposed under-sea mine to be accessed by two deep shafts. It will work the Harbour and Phalen Seams by longwall retreat faces. Annual production is projected at 2 to 3 Mt. Faces will be equipped with double ended shearers, armoured flexible conveyors and powered supports. The distance from the surface to the projected extent of the workings will be 20 km.

Donkin

This will be an under-sea drift mine to be accessed by four slopes working the Harbour, Hub and Lloyd Cove Seams by longwall retreat faces. Annual production is projected at 4 Mt. Faces will be equipped with double ended shearers, armoured flexible conveyors and powered supports. The distance from the surface to the projected extent of the workings is 12 km.

Smaller projects under consideration by other companies include mines at Springhill and Stellarton.

3.5 FIRE CONTROL REQUIREMENTS

Based on past experience and current and projected mining circumstances, the occurence of the following types of fire must be viewed as being possible in Canadian underground mines:

- Open fires in roadways and other accessible locations.
- Open fires on longwall faces.
- Open fires in longwall goaf areas.
- Heatings in rib sides, pillars or other solid coal areas.
- Heatings in mined-out areas.

Inerting techniques appropriate to Canadian circumstances, therefore, must be capable of attacking open fires and preventing or controling heatings.

TABLE 3-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

CANADIAN UNDERGROUND COAL MINES

Province	Operator	Active	Potential*
Alberta	Cardinal River Coal Hinton		Cardinal River Hydraulic
	Luscar Edson		Coal Valley
	Fording Lethbridge		Shaughnessy
	Precambrian Shield Lethbridge		Picture Butte
	Smoky River Coal Grande Cache	#9B Mine	#9G Mine
British Columbia	David Minerals Chetwynd		Chetwynd
	Norco Resources Ltd. Bowron River		Bowron River
	Westar Mining Ltd. Sparwood	Balmer North Balmer South	Baldy Ridge
	Wolf Mountain Coal Nanaimo		Wolf Mountain
Nova Scotia	Cape Breton Dev.Co. Sydney	#26 Mine Lingan Mine Prince Mine	Donkin Lingan-Phalen Glace Bay
	Evans Coal Mines Ltd. Inverness County	Evans Mine	
	Pentum Springhill		Springhill
	Suncor Stellarton		Stellarton

* Under consideration for development within 5-10 years

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SECTION 4 - INERTING SYSTEMS

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.....Cont'd

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SECTION 4 - INERTING MEDIA AND SYSTEMS

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SECTION 4 - INERTING SYSTEMS

4.1 INTRODUCTION

Several different inert gases could be used for fire control, but considerations of availability, cost and other factors have focused the coal industry's attention on three - gaseous combustion products (GCP), carbon dioxide, and nitrogen. Sufficient success has been achieved with these media in many parts of the world to provide a high degree of confidence that one or more will be appropriate for Canadian conditions. Consequently other gases for which no practical experience exists have been excluded from consideration.

This section of the report describes the three principal inert gases used for mine fire control. Each has some advantages and disadvantages that apply in any mine fire control situation and others that apply only in specific circumstances.

The relative merits of inerting media and delivery systems are reviewed and ranked according to their suitability for application in the Canadian coal mining industry.

4.2 GASEOUS COMBUSTION PRODUCTS

4.2.1 General

Gaseous combustion products generally contain a significant proportion of nitrogen along with carbon dioxide, carbon monoxide, water vapour and certain other constituents. GCP are generally low in oxygen and if brought into contact with a fire can reduce its intensity.

Though there are some reported cases of flue gases being conducted into mines for firefighting purposes, any practical method of using GCP must be mobile and provide at the fire site rapid generation of GCP with appropriate chemical and physical properties. This can be achieved by using an inert gas generator (IGG). A number of such machines have been developed of which the most important are the Polish GAG (CMI) series and the Russian GIG series. The Polish GAG machine is available commercially and has been tested by the USBM.

For the purposes of comparing GCP with carbon dioxide and nitrogen only products from the GAG IGG operating in its normal mode underground near a fire area will be considered.

4.2.2 Advantages of GCP

- can be locally generated and are independent of availability of commercial gases
- cheaper than carbon dioxide or nitrogen
- can be delvered to fire zone at rates about of 1800 m^3/min which is much higher than generally possible with carbon dioxide or nitrogen
- can be operational about two hours after delivery of unit to fire site
- carries large volume of water into fire zone to provide cooling.

4.2.3 Disadvantages of GCP

- cannot generally be stored
- IGG needs large kerosene supply
- accurate quality control difficult since composition varies with fuel, oxygen supply and other factors
- constituents are variable and include N_2, CO_2, CO, residual O_2, trace gases, H_2, higher hydrocarbons, toluenes, NO_2, SO_2, etc. as well as carbon spots and water vapour
- obscures monitoring

- rate of generation and delivery difficult to control accurately
- not normally free of oxygen
- delivered into fire zone generally at higher temperatures than CO_2 and N_2
- requires large volumes of cooling water
- needs blower to transmit any distance from point of production
- IGG operation can alter pressure distribution in mine
- high water content can lead to production of hydrogen and/or water gas
- IGG does not comply with flameproof requirements for most jurisdictions.

4.2.4 GCP Delivery Systems

In current practice gaseous combustion products are normally generated and delivered into the fire zone by an inert gas generator (IGG). In most designs the equipment consists of an aero jet engine with afterburner whose products are passed through a cooling system and discharged into the fire area through a duct.

Currently there are two principal types of IGG's available - the Russian GIG (Figure 4-1) and the Polish GAG (Figure 4-2) series. The output of these machines range from the Russian GIG-50 with a capacity of 120 m^3/min to the Polish GAG-3 with a maximum output of more than 1800 m^3/min. A summary of the principal characteristics of these IGG's is provided in Table 4-1.

The composition of the GCP varies according to the type of generator and its operating parameters, though the principal constituents are always nitrogen, carbon dioxide and water. Typical products from a GAG machine have been stated to be composed as follows: .

Constituents	Volume (%) Case 1	Volume (%) Case 2
Water	64.10	63.80
Nitrogen	30.44	30.80
Carbon Dioxide	4.72	5.00
Oxygen	0.74	0.40

Oxygen contents quoted for IGG's ranges from a maximum of 3% for the small GIG-50 to less than 1% for the GAG-3. The exhaust gases generally leave the duct at a temperature between 80°C and 90°C, depending on operating conditions. The primary cooling water can be rejected to waste or can be injected into the evaporative cooler. In this case, the gaseous products are delivered a few degrees hotter than is the case if an external water supply is provided for evaporative cooler.

In a more recent design of the GAG provision is made to use nitrogen for evaporative cooling (Figure 4-3). The nitrogen enables the duct line to be purged before the IGG is put into operation and increases the proportion of nitrogen in the product.

The characteristics of an IGG are a high flow rate coupled with a relatively low delivery pressure. For this reason it is necessary to site the machine at or near the fire zone. The transmission of GCP some seven kilometres through an 800 mm duct has been reported from Poland. Consideration has been given in Germany and Poland to modifying IGG equipment to increase the delivery pressure so that the unit may be installed on the mine surface and the products transmitted to an underground fire through pipes.

4.3 CARBON DIOXIDE

4.3.1 General

Carbon dioxide is widely used in fire extinguishing systems and was an obvious choice when a medium to inert underground mine fires was being sought. It is a colourless gas with a density of 1.98 kg/m^3 STP. It is produced on a commercial scale in most industrialized countries and is available in relatively large quantities. It can be provided in solid form (dry ice) and as a liquid.

Though carbon dioxide has some advantages, the wide availability of commercially produced nitrogen with characteristics more suitable for inerting purposes, has significantly reduced the reported usage of carbon dioxide.

4.3.2 Advantages of Carbon Dioxide

- can be stored and transported as a liquid or solid
- is an extinctive gas
- cools the fire area.
- available in commercial quantities
- can be evaporated in commercial mobile plants.

4.3.3 Disadvantages of Carbon Dioxide

- chemically highly reactive
- absorbed by water
- density significantly greater than air
- is a noxious gas
- is an irritant
- can be reduced by hot carbon to carbon monoxide
- interferes with carbon dioxide monitoring and trend interpretation
- if reduced to CO interferes with carbon monoxide monitoring and trend interpretation
- more expensive than nitrogen or products of combustion
- makes interpretation of the Coward diagram difficult

- corrosive if transmitted in steel pipelines in the presence of moisture
- can raise the carbon dioxide concentration above statutory limits
- in practice cannot be delivered to fire zone in gaseous form at more than about 500 m^3/min per evaporating unit.

4.3.4 Carbon Dioxide Delivery Systems

In some mine fire combat situations solid carbon dioxide has been transported to the fire zone and been placed on the fire. In other cases bottled carbon dioxide has released directly into the fire zone or liquid carbon dioxide has been directed onto the fire through boreholes from the surface or released into the air stream or through stoppings.

Should a large continuous flow of carbon dioxide be required then commercial evaporation units can be employed similar to those described below for nitrogen evaporation.

4.4 NITROGEN

4.4.1 General

Nitrogen, which is one of the principal constituents of air is an inert gas with a density of 1.25 kg/m^3 STP. It is stable and can be stored or transported in liquid form. Nitrogen is available commercially and a number of nitrogen contracting companies provide specialized nitrogen services to the oil, gas and chemical industries. Nitrogen technology is well developed and is being increasingly applied for fire control in the mining industry.

4.4.2 Advantages of Nitrogen

- inert and non-reactive with carbon or other materials found in a mine
- not noxious
- density close to that of air
- stable
- readily available in large quantities
- non-corrosive
- safe to handle
- is an extinctive gas
- generally can be supplied completely free of oxygen
- no fumes
- does not interfere with monitoring
- can be evaporated in commercial mobile plants
- cool the fire area
- can be stored or transported in liquid form.

4.4.3 Disadvantages of Nitrogen

- subject to evaporation losses when stored and expensive to stock
- more expensive than products of combustion
- in practice cannot be delivered to fire zone in gasious form at more than about 300 m^3/min per evaporation unit.

4.4.4 Nitrogen Delivery Systems

Nitrogen can be delivered to a fire zone as a liquid or in gaseous form. The four principal systems that have been used in the combat of mine fires are:

Liquid nitrogen	delivered by tanker into fire area
Gaseous nitrogen	produced by air separation unit piped to fire area
Gaseous nitrogen	produced by ambient temperature evaporation unit piped to fire area
Gaseous nitrogen	produced by externally heated evaporation unit piped to fire area.

The volume of nitrogen required to handle the emergency is a major factor in selecting the delivery system. Using currently available commercial equipment gaseous nitrogen can be delivered at up to about 50 m^3/hour by an ambient temperature evaporation unit, up to about 300 m^3/hour by an externally heated evaporation unit and up to 13 m^3/hour from an air separation unit.

Where large flow rates are required only externally heated evaporators currently provide sufficient capacity and in most cases mine fires have been fought with units of this type. The evaporators have generally been propane or diesel heated, or hot water has been provided from the mine hot water system or a locomotive boiler.

A pre-requisite for the delivery of liquid nitrogen by underground tankers is the accessibility of the fire zone to the mine transport network.

Typical delivery systems currently in use in the mining industry comprise:

a. Liquid Nitrogen by Underground Tanker

The introduction of liquid nitrogen into a fire zone has been reported from Poland where special vehicles for underground transportation of nitrogen have been designed and constructed. In this system (Figure 4-4) liquid nitrogen is filled into insulated pressure vessels from road tankers. The vessels are mounted on mine car chassis for transportation on the mine rail system. The tanker units each hold approximately 0.3 m^3 of liquid nitrogen which yields approximately 207 m^3 of gaseous nitrogen. In order to provide continuity the vessels are arranged in sets of six to eight. These are connected to a release system so that a continuous flow of nitrogen can be obtained until the set is exhausted. Because of mine logistics it is generally necessary to provide spare sets of vehicles for transportation and refilling on the surface. In order to maintain one set at the fire, at least two additional sets would normally be required.

Because the nitrogen is released and vapourized at the fire site, it reaches the fire zone at a low temperature and the cooling effect is greater than that attained with gaseous nitrogen piped from the surface. However, the system is intermittent and of limited capacity.

b. Ambient Temperature Evaporation Units

Evaporation units of this type have limited capacity, but by drawing most of the evaporation heat from the atmosphere are lower in cost to operate than evaporation units with external heat sources. Ambient evaporators are well suited to long-term low-volume nitrogen production, for example, as required in some spontaneous heating control applications.

A typical ambient temperature evaporation unit (Figure 4-5) consists of a set of individual evaporators with large surface areas in which the liquid nitrogen is vapourized by raising its temperature to ambient. The product gas is generally put through secondary heating systems to raise the outlet temperature to between 10°C and 20°C to prevent freezing of the delivery pipes. The secondary heaters are commonly electrically powered water bath or air heaters. The output of a single group of ambient evaporation unit is typically 40 to 50 m^3/hour. The principal advantages are the low power costs, minimal maintenance and availability of remote control that results in low manpower requirements. These units typically take three days to install and activate and are best suited to long-term inerting rather than emergency response.

c. Externally Heated Evaporation Units

The principal method of delivering gaseous nitrogen to mine fires is by evaporating liquid nitrogen on the mine surface using an external heat source and transmitting gaseous products via pipelines to the fire area. Evaporating units of this type have been developed in France, Germany, Poland, USSR and other mining countries that operate on the same basic principles, but differ in detailed design. Until recently the main categories of externally heated apparatus were:

i direct propane heating of water bath
ii indirect oil fired heating of water bath.

Some principal features of these units are given in Table 4-2.

A recent development is the all electric vapouriser.

Type ii evaporators are widely employed and exemplified by the Messer-Griesheim propane unit (Figure 4-6) and Nowsco diesel unit (Figure 4-7). The electric vapouriser obtains much of its energy from the air, supplemented by electric heating (Figure 4-8). The capacities of externally heated evaporators for mine fire control are, to some extent, limited by the need to make the units transportable on conventional sized trailers. Clearly a stationary plant could be constructed of larger capacity than mobile plants.

When a single evaporating unit is unable to provide the total nitrogen requirements additional units may be connected in parallel to increase the flow rate.

To ensure continuous delivery it is normal to provide liquid nitrogen storage capacity at the evaporator site generally in the form of a mobile tank, although some mines provide permanent storage tanks.

d. Air Separation Method

Production of nitrogen from air is relatively simple since nitrogen exists in air in molecular form. The principal methods are the pressure swing (absorption) process using a carbon molecular sieve and the refrigeration process. Although the absorption process is preferred for small and medium quantities, the refrigeration process (Figure 4-9) combines favourable energy consumption with a simple process and thus a compact unit.

In a separation unit using the refrigeration process, the main components are an air compressor, separator and filters for precleaning the compressed air, heat exchanger for cooling down the process air and removing water and carbon dioxide, absorber for removing hydrocarbons, expansion turbine for refrigeration, heat exchanger for condensing the air and rectifying column for separating the air.

The GAN 800 M air separation unit built by Linde AG uses the refrigeration process. The unit has a rated output of 13 m^3/min and is divided into a machine section (engine, compressor, oil cooling system) and an insulated refrigeration section (cold box). Each section is accommodated in a 20' container mounted on a semi-trailer and can be hauled in normal road traffic by a tractor truck. Comprehensive measuring and control systems make the running of the process almost automatic.

4.5 RANKING

GCP produced by an IGG can be delivered into a fire zone at high flow rates and the response to fire emergencies can be rapid. However, in their present form IGG's have the disadvantage of requiring large quantities of kerozene near the fire zone and would not meet Canadian regulatory standards for underground flame proofing. For this reason they are not suitable for general application in Canadian underground coal mines.

In specific cases the IGG could have an application, for example in a small coal mine where the unit could be set up on the surface to inertize the whole mine. In non-gassy mines (i.e. some coal mines and most non-coal mines) the IGG could be considered for use underground.

Carbon dioxide has some advantages (paragraph 4.3.2) but its numerous disadvantages (paragraph 4.3.3) make it unsuitable as an inerting medium. In particular the interference of carbon dioxide with ventilation monitoring is a major drawback. Carbon dioxide is more expensive than nitrogen. Since nitrogen is readily available for use at Canadian coal mines, carbon dioxide is a less favourable alternative.

Nitrogen is the most suitable inert gas for mine fire control in Canada. It has many advantages (paragraph 4.4.2) and relatively few disadvantages (paragraph 4.4.3). Its effectiveness and safety have been demonstrated in a large number of fire control incidents. The gas is available in adequate quantities in both eastern and western Canada and a strong Canadian capability in nitrogen technology has been developed through experience in the oil and gas industry.

Nitrogen inerting is the preferred method for general application in Canadian underground coal mines.

4.6 RECOMMENDATION

Canadian underground coal operators should prepare for the use of nitrogen for mine fire control in appropriate circumstances. Contingency plans should be drawn up for combatting mine fires and appropriate provisions should be made to permit the rapid deployment of nitrogen to any part of the mine affected by fire. Arrangements should be made with nitrogen suppliers for the supply and delivery of nitrogen in adequate quantities for mine fire emergencies.

TABLE 4-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

INERT GAS GENERATORS

Parameters	GIG-50*	GIG-4*	GIG-1500*	GAG-2**	GAG-3+
Productivity, m^3/min	50-120	340	800-1500	720-1500	850-1870
Oxygen content, % vol.	2-3	1-2	1-2	1	1
Fuel consumption, kg/h	260	820	3400	2880	1080^{++}
Water consumption, m^3/h	8	27	50	30	69
Dimensions, m					
Length	4.0	6.5	11.2	11.9	15.0
Height	0.7	0.8	1.0	1.0	0.9
Width	0.7	0.9	1.1	1.1	1.1
Supply voltage, V	280/660	Self-powered	Self-powered	24	24

* Source: Khudosovtsev and Kozlyuk
** Source: Villedent
+ Source: Kopex brochure
++ Value provided in Kopex brochure appears anomalous

TABLE 4-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

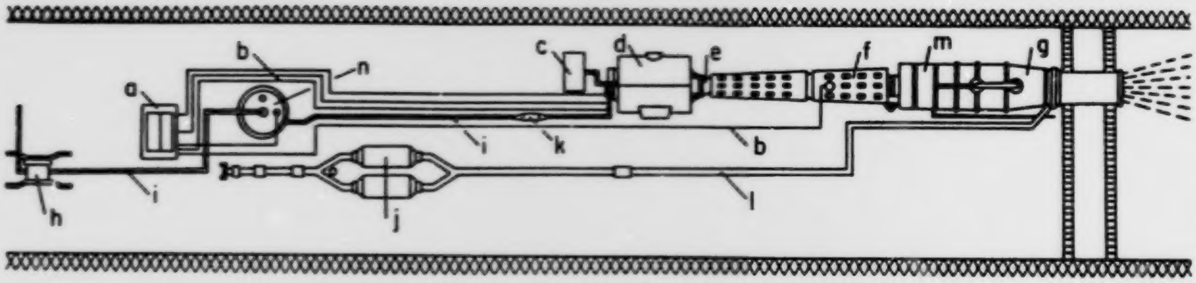
NITROGEN EVAPORATION UNITS

Country	Germany	Germany	France	UK	Germany	USSR
Make	Messer Griesheim	Linde	Air Liquide	Nowsco	Linde	N/A
Model	MoGruV	N/A	N/A	MEV	GAN 800	GGR
Type	WBI	WBI	WBI	E/A	AS	AS
Capacity (m^3/min)	125	300	167	120	13	100
Power Source/ Consumption	Propane 66 kg/hr	Diesel	Diesel	Electric	Diesel 63 kg/hr	N/A 12000 kW/hr

WBI = Water bath indirect
E/A = Electric/ambient
AS = Air Separation

FIGURE 4-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

INERT GAS GENERATOR GIG-4

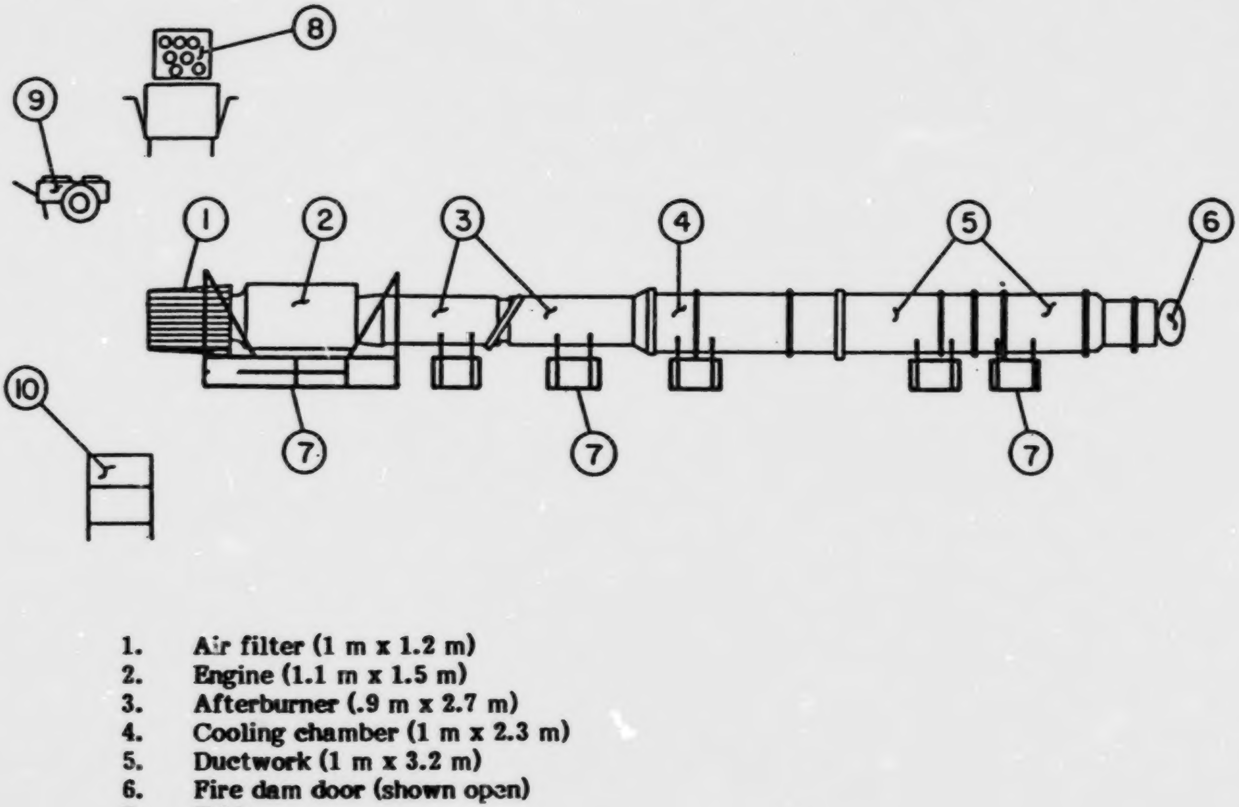


a.	operator's module
b.	cable
c.	aircraft generater and battery
d.	turbine
e.	air injector
f.	kerosene evaporater
g.	cooling chamber
h.	kerosene pump
i.	kerosene hose
k.	kerosene filter
j.	water filter
l.	water line
m.	after burner
n.	kerosene container

FIGURE 4-2

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

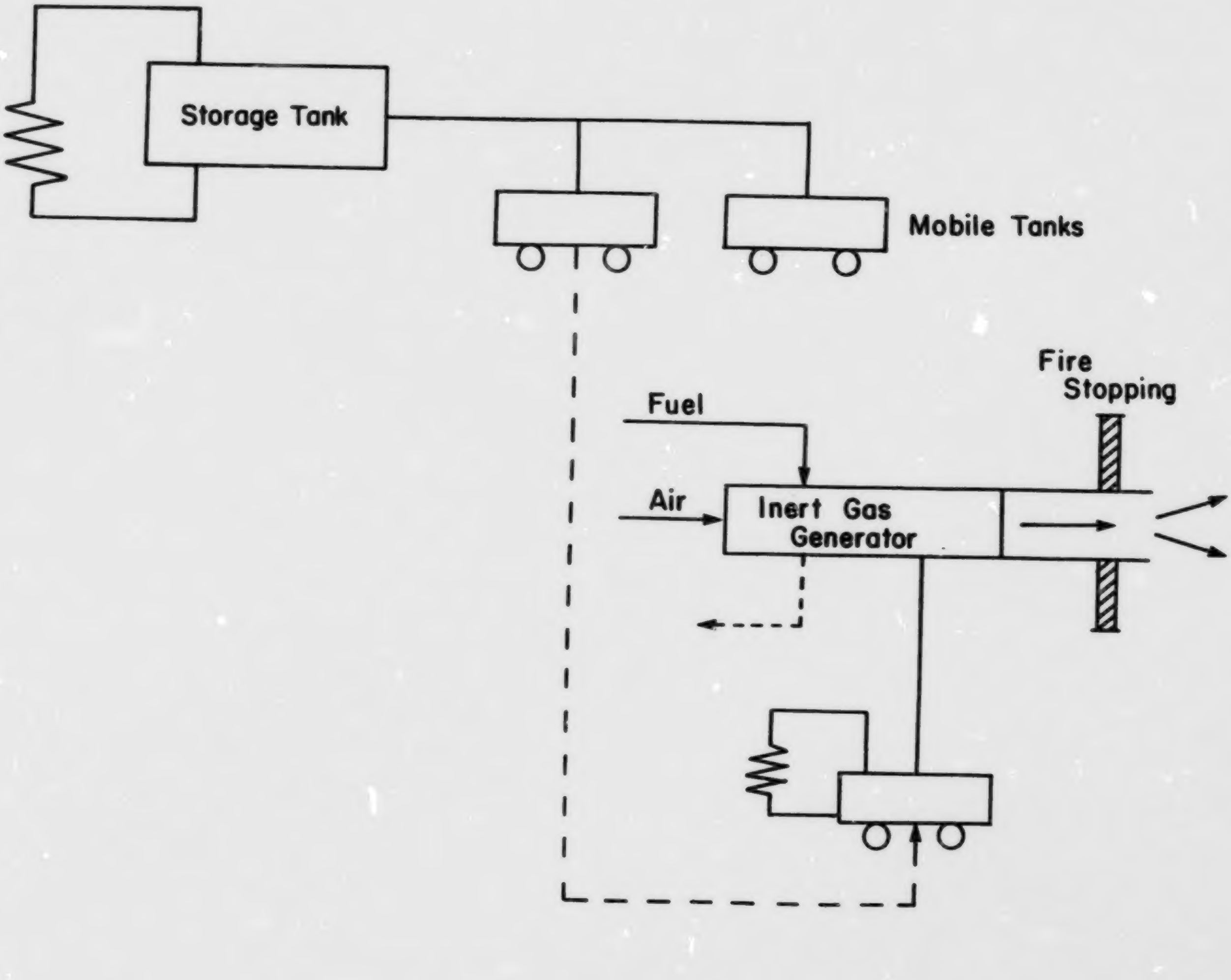
INERT GAS GENERATOR GAG



1. Air filter (1 m x 1.2 m)
2. Engine (1.1 m x 1.5 m)
3. Afterburner (.9 m x 2.7 m)
4. Cooling chamber (1 m x 2.3 m)
5. Ductwork (1 m x 3.2 m)
6. Fire dam door (shown open)
7. Skids
8. Control console
9. Battery cart
10. Fuel tank and pump

FIGURE 4-3
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

EVAPORATION OF LIQUID NITROGEN INSIDE INERT GAS GENERATOR



AR-72

FIGURE 4-4
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

APPLICATION OF LIQUID NITROGEN

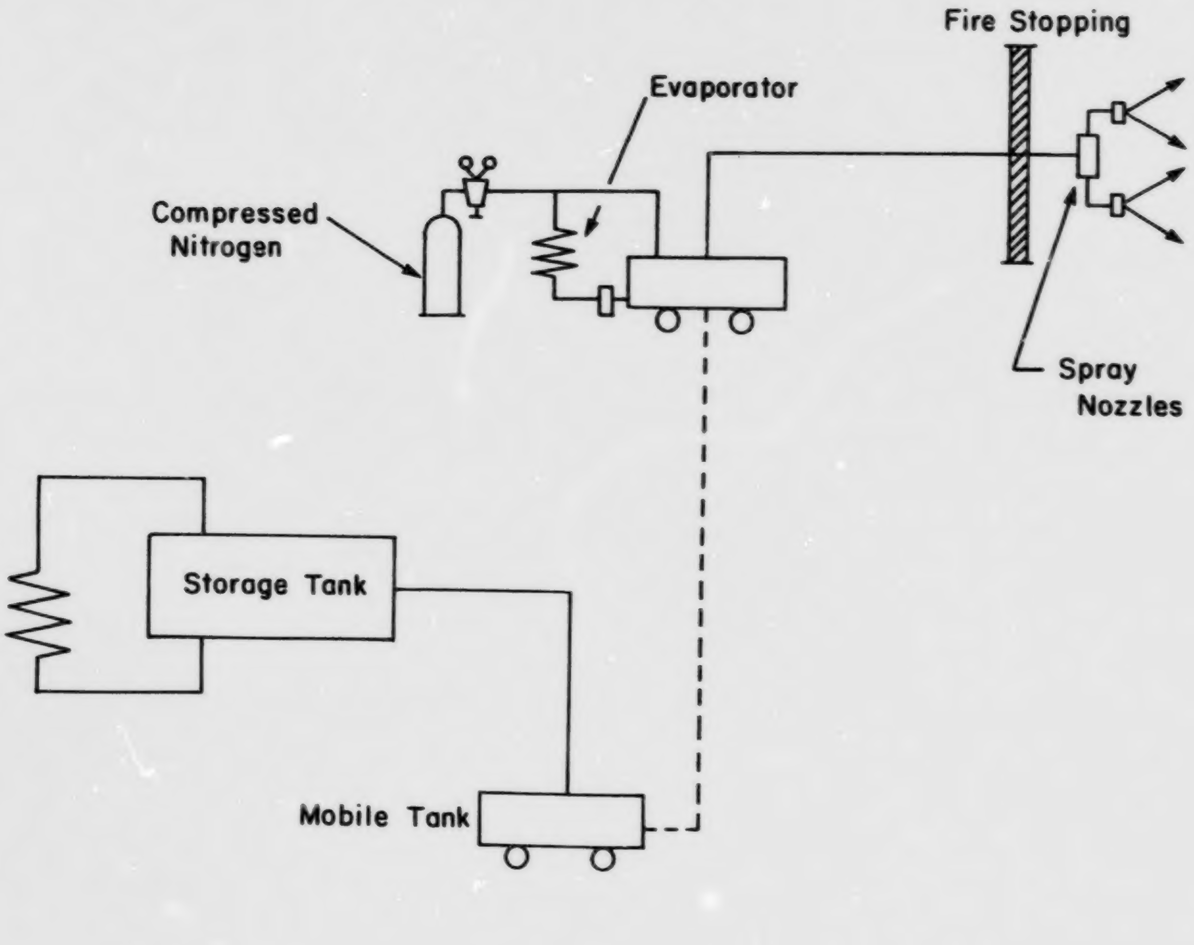


FIGURE 4-5
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

AMBIENT EVAPORATOR

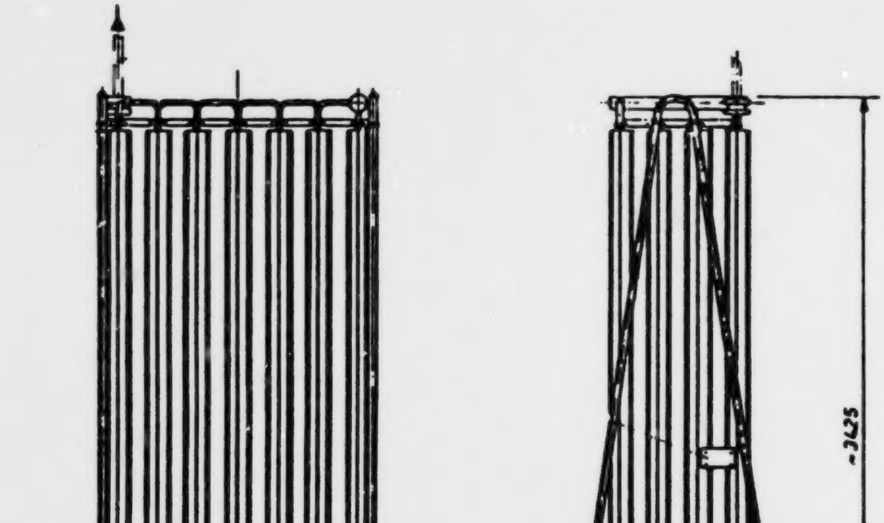


FIGURE 4-6

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

MESSER-GRIESHEIM NITROGEN EVAPORATION UNIT

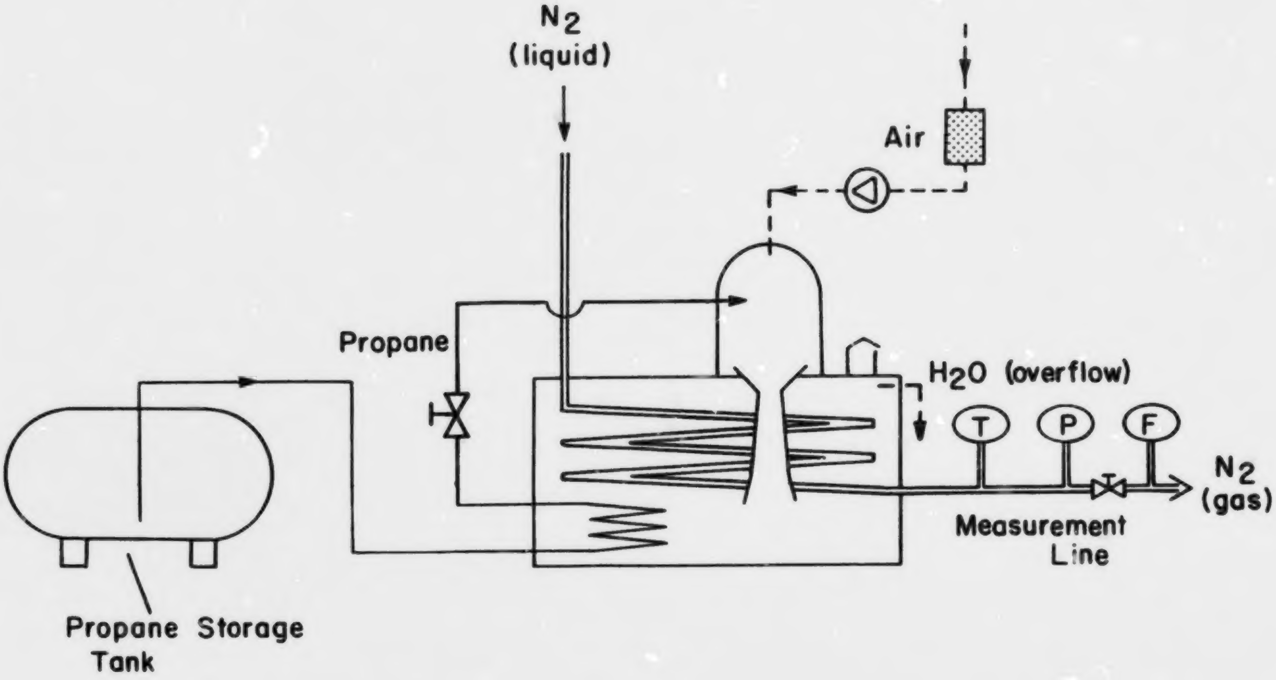


FIGURE 4-7

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

NOWSCO DIESEL HEATED EVAPORATION UNIT

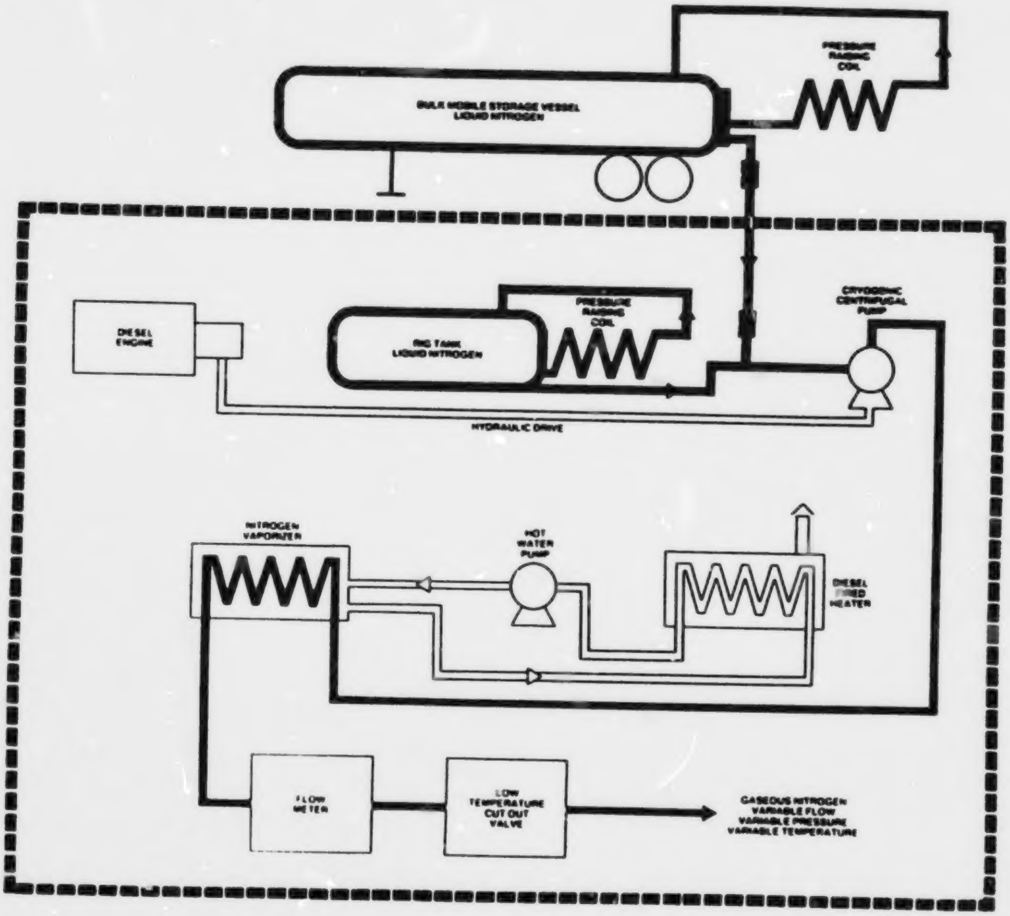


FIGURE 4-8

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

NOWSCO MOBILE ELECTRIC VAPORIZER

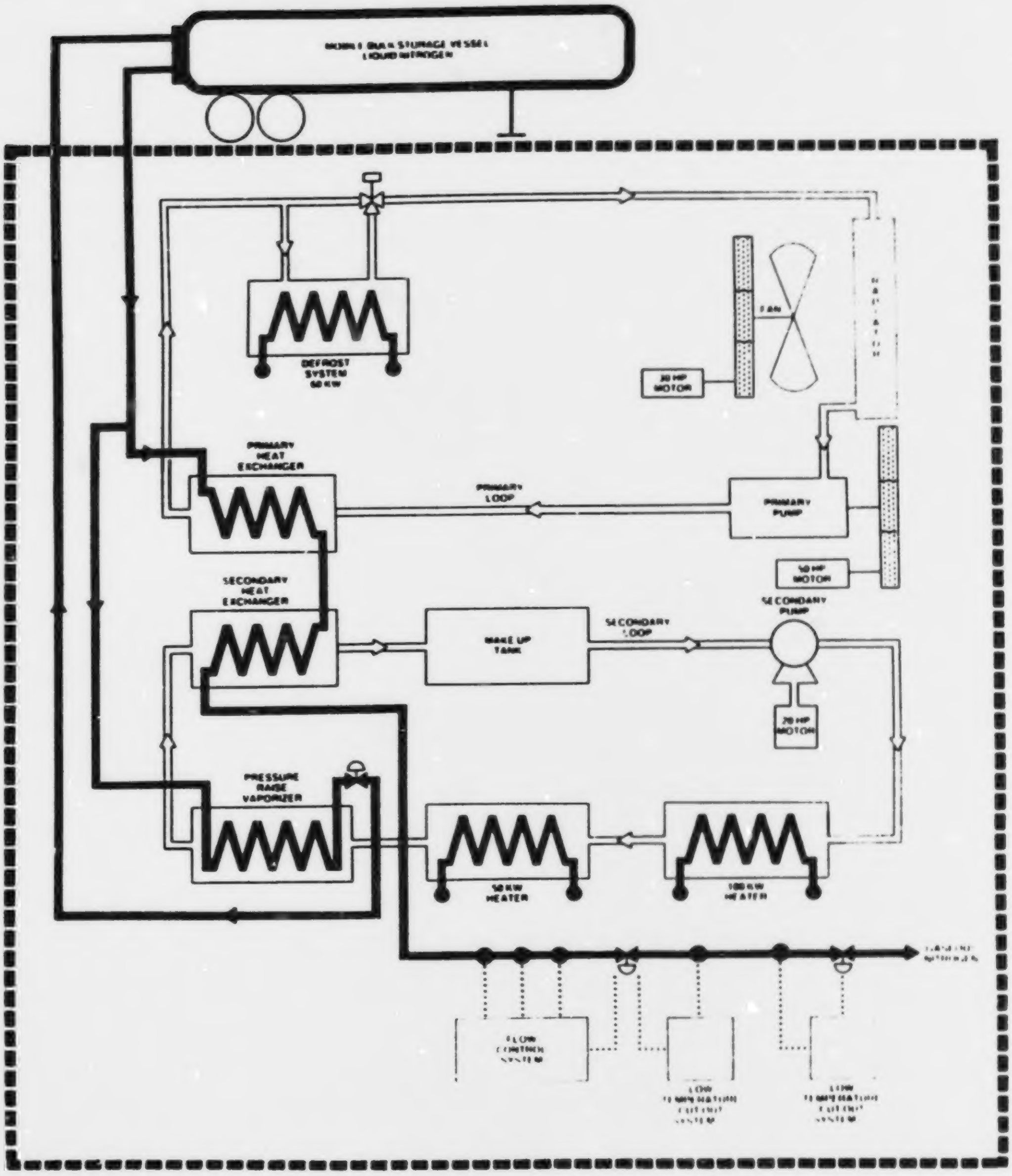
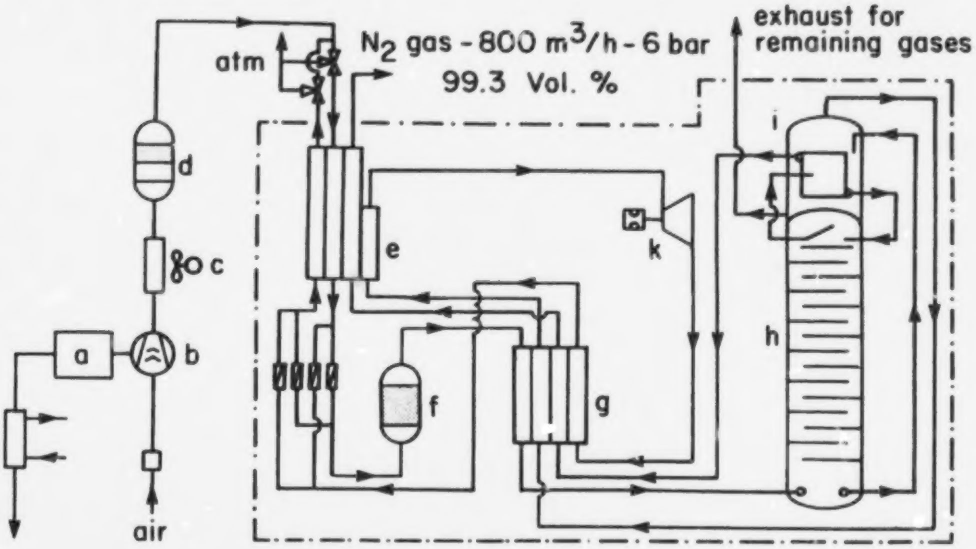


FIGURE 4-9

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

LINDE GAN 800 AIR SEPARATION UNIT



a. diesel motor
b. compresser
c. air cooler
d. oil filter
e heat exchanger (H_2O + CO_2 barrier)
f. gas adsorption (C_2H_2)
g. liquifier (-170°C)
h. air separation column (O_2 from N_2)
i. condenser (N_2 liquifaction)
j. expansion turbine (refrigeration)

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SECTION 5 - APPLICATION OF INERTING IN CANADIAN MINES

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5-1	Surface Connection for Nitrogen
5-2a/b/c	Flow of Nitrogen
5-3	Manifold for Nitrogen Distribution
5-4	Example of Surface Plan of Inerting Arrangements
5-5	Example of Underground Plan of Inerting Arrangements
5-6	Relationship Between Airflow, Nitrogen and Oxygen

SECTION 5 - APPLICATION OF INERTING IN CANADIAN MINES

5.1 RECOMMENDED SYSTEM

Nitrogen inerting has been widely and successfully applied and can control a range of mine fire situations.

The inerting system recommended for general application in Canadian underground coal mines is to use gaseous nitrogen vaporized from commercial liquid nitrogen on the mine surface and piped underground to the fire zone. This technique is flexible and can be used to deliver low to high flow rates.

In this section practical aspects of the preparation and application of nitrogen inerting in emergency conditions are discussed. The Codes of Practice developed by the Haupstelle fur das Grubenrettungswesen (Germany) have been a valuable source of information.

Although nitrogen is recommended as the best technique for Canadian conditions, the application of other inerting methods may be appropriate in certain circumstances.

5.2 EQUIPMENT AND MATERIALS

The principal elements of a nitrogen inerting system are:

- nitrogen evaporation plant
- pipeline to fire zone
- measuring equipment

Although equipment and material requirements will differ according to the specific circumstances at a mine the following general considerations will apply in the majority of cases.

Some equipment and materials required for inerting are available on a rental basis from specialized nitrogen contractors, while other parts of the system will have to be purchased by the mining company and installed or held in readiness at the mine site.

Nitrogen Evaporation Plant

Mine operators should make provisions for a nitrogen evaporating plant to be available on short notice comprising an intermediate nitrogen storage vessel, an evaporation unit and appropriate monitoring and regulating systems. This equipment is offered by several Canadian nitrogen contractors. Mine operators should review availability, specifications and costs from appropriate sources and enter into a formal agreement with a supplier.

Based on extensive experience offshore, the requirements for an evaporation plant for mine fire control are:

- throughput of gaseous nitrogen 250 m^3/min or more
- delivery pressure suitable for mine pipes (generally 150 psi)
- delivery temperature in a range high enough to prevent freezing (generally 10°C - 20°C)
- mobilization time to site not more than six hours
- capability to deliver continuous supply of nitrogen at 250 m^3/min for a period of five days and thereafter at reduced rates if necessary for a period of months.

These parameters are provided as a guide and can be modified in the light of circumstances at a mine.

Pipeline to Fire Zone

The surface evaporating plant must be connected to the fire zone underground either by an additional pipeline exclusively for nitrogen or by utilizing some of the existing mine pipe networks or a combination of these two approaches. There are cost and time advantages in using existing pipework but if this is not possible additional nitrogen pipework will have to be installed.

In mines with a high fire risk it may be desirable to install pipelines dedicted exclusively to nitrogen supply. In other cases plans should be made to adapt existing pipe ranges or to run temporary pipelines in the event of an emergency.

The principal elements of the pipe system comprise:

- connection to evaporating unit
- intermediate pipework
- delivery

Pipes normally used for nitrogen transmission are 100 mm diameter or larger and capable of withstanding pressures in the range of 100 - 150 psi. For long transmission distances large diameter pipes are advantageous.

Connection to Evaporating Unit

A permanent connection should be provided on the mine surface to which the nitrogen evaporation plant can be attached. This connection should be located at a site with space to accommodate the nitrogen storage vessel and evaporating unit and should be equipped with mine and external telephone and appropriate water and electrical services. The site should be selected so that waste gases or nitrogen do not pollute the mine intake air.

A design for permanent nitrogen connection point which has been adopted in France is shown in Figure 5-1. This layout can be modified to suit conditions at a particular mine.

Intermediate Pipework

The largest diameter pipes possible should be used to minimize pressure losses both in the shaft and underground. The minimum nominal diameter should be 100 mm and there should be no cross-sectional constrictions.

Water and compressed air are frequently essential for mine rescue purposes such as fire extinguishing or sealing off. In some cases the rescue team can be supplied with compressed air and water via flexible hoses and the compressed air or water pipes released for inerting.

The amount of nitrogen that can be conveyed through lines of various diameters, lengths and pressures is given in Figure 5-2a, 2b and 2c.

If it is intended to run an additional pipeline for nitrogen, the preparatory work should include a survey of the mine to determine the optimum route and preparation of any clearances or adjustments to the pipe layout that may be required. Stock levels for rigid or flexible pipes for emergency use should be assessed and the required amounts provided on the mine surface.

The 150 mm flexible pipe widely used in the German coal industry has proved very successful in practice. Some modifications to the German specification could be made to suit Canadian circumstances and product availability. This type of pipe requires special clamps for support in shafts.

Where it is intended to convert existing pipe networks to carry nitrogen a survey must be made to determine which pipes will be used and how they can be converted and isolated for nitrogen transportaton. Provisions should be made to facilitate clearing out compressed air, water or other materials from the pipes and to enable rapid closing off of branches to direct the nitrogen to the fire zone. In certain cases some preliminary local modifications of the pipe network will facilitate rapid conversion to nitrogen. Remotely operated valves may have an application, though the possibility of the interruption of the electricity supply in an emergency must be considered. All pipes used for nitrogen must be capable of the duty to be performed in terms of both temperature and pressure.

Delivery

Where flexible pipes are used the range should terminate in a steel pipe secured to reduce the risk of lashing. Before the nitrogen range is connected to the fire zone it must be purged of air. This can be achieved by inserting a suitable T-piece near the end of the nitrogen range.

During firefighting the nitrogen will either be delivered through a stopping or into injection pipes or lances set in boreholes. Connections to stoppings should include takeoffs for measuring nitrogen flow and pressure. Where localized nitrogen injection is being carried manifolds should be provided to facilitate connections to the individual injection pipes and the attachment of measuring devices. A typical manifold design is shown in Figure 5-3.

Measuring Equipment

When combatting a mine fire with nitrogen equipment must be available to measure the flow of nitrogen and to monitor the mine environment. As a minimum the following equipment should be on hand:

- hand-held methane meters
- hand-held oxygen meters
- indicator tubes for carbon monoxide
- indicator tubes for hydrogen.

In a fire emergency it is imperative that proper air analysis facilities are available at the earliest possible time to provide accurate assessments of methane, carbon monoxide, carbon dioxide, oxygen and hydrogen. These facilities should be provided for each mine or on a shared basis. If an individual mining company does not have its own facilities, arrangements must be made to ensure coverage in the event of an emergency.

In mines with a high risk of spontaneous combustion, continuous monitoring facilities should be provided. These can provide an early indication of an incipient fire and enable remedial action, which may include inerting, to be initiated before the fire gets out of control. Continuous monitoring during and after inerting enables the application of nitrogen to be optimized.

Hand-held instruments, indicating tubes, air analyzer and continuous monitoring systems are available from a number of Canadian suppliers.

5.3 PROCEDURES

5.1.1 General

Inerting may be used in a range of circumstances including:

- through seals to control a fire or reduce explosion risk
- without seals to control a fire or reduce explosion risk
- into solid coal or a mined-out area to control a localized fire or heating
- into a mined-out area to prevent the onset or development of spontaneous combustion.

The decision whether to use inerting to complement or substitute for other means such as direct attack or stopping off, should be made by mining and ventilation engineers after consideration of the relevant factors including safety, logistics and economics.

An inerting operation has the following stages:

- decision to inert
- preparation for inerting
- commencement of inerting
- follow-up measures.

The decision to inert will require an assessment of the flow rate of nitrogen. This will be a function of the capacity of the evaporator or evaporators and of the pipework available to transport the nitrogen. The air flow passing by the fire zone must be assessed and the decision made as to the maximum or minimum air required depending on the extent of any methane. The sites and nature of any stoppings must be determined.

The operation commences with ordering the nitrogen contractor to site and preparation of the pipe route to the fire zone. A reconnaissance should be made to confirm the exact location and construction method for any stoppings. The measuring equipment for the control of nitrogen and air quality should be established. The air quantities flowing in the circuits that may be effected by the operation should be checked.

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When inerting is to be put into operation two circumstances are possible. In the first, the stoppings are completed before the nitrogen is ready. In this case the nitrogen would be connected up to deliver through the stoppings and would be released at the determined rate. In the second case the nitrogen supply is available before stoppings are completed. In this case it may be beneficial to commence nitrogen flow immediately to check the effectiveness of the various elements in the system and to increase the nitrogen content in the air flowing towards the fire with the possibility of reducing the fire intensity or the risk of explosion.

The inerting phase begins when the nitrogen flow is started and continues until the whole area is filled with nitrogen and the oxygen reduced to a level that will not support combustion or a localized fire is brought under control.

The follow-up phase includes reducing leakages and completing the sealing of the return to prevent air ingress by counter circulation. The sealed area should be topped up until the carbon monoxide levels or submicron particulate measurements indicate that the fire has been completely extinguished. It may be necessary to wait for some time after the fire has been extinguished before a sealed area is cool enough to be re-entered. Cases have been reported of circulating the contents of a sealed area through a cooling system to reduce the temperature as rapidly as possible. It must be recognized, however, that considerable expertise is required to conduct such an operation.

5.3.2 Surface Plant

Plans should be prepared and regularly updated showing the arrangements on the surface (Figure 5-4) and underground (Figure 5-5) pipe routes to potential fire locations.

The quantity of nitrogen required for inerting in a particular case depends on a number of factors such as the location of the fire - whether it is open or deep seated - the volume of air passing the fire zone, the point from which the nitrogen is to be applied and whether it is to be applied in the presence of normal ventilation or into seals.

The relationship between liquid and gaseous nitrogen quantities can be estimated from the following table:

Gaseous (15°C: 1 bar) (m^3)	Liquid (-195,85°C: 1 Bar) (litre)	Weight (kg)
1.	1.448	1.170
0.691	1.	0.808
0.855	1.238	1.

The physical capacity of the available nitrogen evaporation plant determines the maximum possible flow rate of nitrogen.

The nitrogen deliverable to a fire depends on the diameters of the pipe network. Where the pipe diameter from the evaporator to the delivery point is the same throughout the flow can be calculated from simple straight line charts (Figure 5-2a, 5-2b and 5-2c). Where ranges consist of various pipe diameters it is advisable to preprogram calculations into a computing device so that flow rates to any fire site can be calculated rapidly.

Mobile liquid nitrogen plant is usually heavy and axle loads of up to 10 tons may be anticipated. Vertical clearances adequate for the nitrogen equipment must be provided. The liquid nitrogen is maintained at very low temperatures close to -200°C. A typical nitrogen tanker with a nominal capacity of 38 tons carries the equivalent of 17 000 m^3 of gaseous nitrogen at 15°C and one bar pressure.

Double walled pressure vessels constructed from low temperature steel and transported on semi-trailers, act as intermediate capacity between the tanker and the evaporation plant. A typical unit is 4 m high, 2.5 m wide and 13 m long.

The pipes used to connect the tanker to the storage vessel must meet low temperature specifications. The temperature of the inert gas should be between 10° and 20°C and never fall below +5°C in order to prevent icing problems.

A temperature measuring device which gives an alarm when temperatures exceed or fall below predetermined values should be provided.

The inert gas plant should be supervised only by trained personnel conversant with the safety regulatons and with the operation of the plant who have been informed of the risks involved in handling liquid and gaseous nitrogen and of the measures to be taken in case of accidents or malfunctions.

Liquid nitrogen should be transferred from tank trucks only when the vehicles are parked on level ground. Before the fuel lines are attached vehicles must be secured against movement.

When transferring liquid nitrogen, operating personnel must wear protective gloves and goggles. The liquid should on no account be allowed to splash onto human skin. Breathing apparatus must be available.

The area around the inert gas plant in which splashing of liquid nitrogen is possible must be temporarily cordoned off and sign-posted during transfer. It may be entered only by the operating and supervisory personnel.

Persons whose skin has come into contact with liquid nitrogen should be treated as suffering from burns. The parts of the body affected should be covered with sterilized cloths. The victim must drink as much tea or water as possible and be taken immediately to hospital.

The nitrogen line running to the shaft must be sited so it cannot be damaged by manoeuvering vehicles. Pipe or cable conduits can be used for this purpose. A measuring section should be placed in the nitrogen line and the recording or measuring devices located, if possible, in a heated room.

An emergency cut-off valve should be provided between the evaporator and the connection to the mine pipe range. The equipment should be electrically grounded.

5.3.3 Nitrogen Transmission

A procedure should be set up for the purging of the inert gas pipelines. This must be carried so as to prevent the introduction of oxygen into the fire zone. In normal circumstances the end of the nitrogen range or a T-piece would be positioned clear of the fire zone during the flushing process. The effectiveness of the purge should be checked by measuring the oxygen flowing through the pipe. This must be reduced to zero.

Equipment to measure pressure temperature and flow rates should be provided at the connection to the main pipe range with two reversible ranges (0 to 150 and 0 to 300 m^3/min). A continuous recording instrument should be available for use in an extended inerting operation.

A set of regulating equipment must be provided at the inbye end of the inerting line.

The pressure must be monitored to ensure that the safe operating pressure for the pipelines in use is not exceeded and if possible the line should be fitted with excess-pressure or pressure-limiting valves. Typical pressure limits for mining pipes are:

Compressed-air pipes	maximum operating pressure 10 bars
'High pressure' water pipes	maximum operating pressure 25 bars
Methane drainage pipe	maximum operating pressure 4 bars

The flow and pressure of nitrogen should be measured at each branch in the delivery line.

Direct voice communication should be provided between the surface inert gas plant and the underground delivery point.

Underground inerting lines should be regularly inspected to check for leakage and to ensure that oxygen levels remain over 18%. Access to roads carrying nitrogen pipelines should be restricted.

5.3.4 Inerting Procedure

Adequate supplies of liquid nitrogen must be sustained and where necessary, reserve evaporators should be available.

Care must be taken to ensure that nitrogen pipes are not damaged by the movement of the cage in the shaft or by haulage and transport activities in roadways.

The evaporator plant must have monitoring facilities for volume, flow, temperature and pressure and readings should be communicated to the incident control room. It is essential to monitor pressure underground and when several discharge points are in operation, flow rates must also be observed.

To check the effectiveness of inerting and the composition of the atmosphere in the fire zone, air must be regularly monitored. Recording instruments are particularly effective for methane, carbon dioxide and, in particular oxygen, whose values can be communicated to the control room. Compact measuring instruments are being developed for this purpose. In certain cases hydrogen concentration can also be monitored. Monitoring by measuring instruments can be checked by additional gas sampling. When injecting into a goaf area the measuring instruments can be connected directly to the return side relief pipes.

To avoid the danger of insufficient oxygen, the flow of inert gas must be stopped if the main fan should fail. Nitrogen pipes must be regularly inspected for leaks.

Sufficient portable gas monitors should be available to those who may be at risk from fire gases or lack of oxygen. Workings in which the fracture of a nitrogen pipe can reduce the oxygen concentraton to under 18% by volume, must be isolated or manned only by rescue personnel. Other personnel may be equipped with oxygen self-rescurers as escape apparatus but the work should only be conducted under the supervision of rescue personnel.

All compressed air equipment should be run on nitrogen to minimize the risks associated with oxygen from the exhaust air.

When mine rescue teams are working in breathing apparatus in an inert gas atmosphere, all normal precautions for operating in an oxygen deficiency should be observed and teams should be thoroughly briefed on the hazards. Some authorities recommend that they carry additional oxygen self-rescurers and that resucitation equipment be available at the worksite.

Where fires have occurred in accessible zones the use of inerting can reduce the seriousness of the fire and the production of carbon monoxide so the areas can be sealed off without the use of breathing apparatus or the fire can be extinguished by injection of bentonite or similar materials.

Where inerting is used to control a fire behind seals the object is to reduce the oxygen concentration to less than about 4% by volume and preferably to negligible amounts. In these conditions methane and air mixtures are not explosive. Where hydrogen is present, the oxygen level must be reduced further.

In some cases limited nitrogen generation capacity may make it necessary to consider restricting the air flow to the fire zone to permit sufficiently high concentrations of inert gas to be applied to the fire. However, it is essential that no alteration be made to the ventilation flowing through a fire area without careful consideration of the implications by skilled ventilation engineers.

To achieve the necessary reduction in oxygen level, nitrogen must be introduced in the correct quantities. Any sealing off operation must be carefully co-ordinated with the nitrogen flow. If nitrogen can be delivered early in the process, it may be possible to preempt the explosion risk.

When inerting starts the natural oxygen nitrogen balance in the air flowing to the seat of the fire is changed. The fire intensity index trend is no longer valid and changes in the state of fire must be interpreted by calculating the volume of CO produced from measurements of the CO content in the return air. It is essential that a continuous supply of nitrogen to the underground fire zone is maintained.

In certain cases it is possible to use light stoppings on the intake side to reduce the air flow sufficiently to enable the necessary balance to be achieved between air flow and nitrogen quantities available. In certain cases no intake stopping may be required. Once inerting has proved successful the intake stoppings can be constructed in the most appropriate positions.

If explosion proof stoppings are required, normal firefighting practice requires that an area be sealed, should continue to be ventilated until the stoppings can be closed simultaneously. For this reason large diameter specially designed ducts should be provided through stoppings. In a sealing off operation, ventilation should be continued until the seals can be closed and nitrogen injection started.

Explosion proof seals should only be removed after air analysis indicates that the fire is completely extinguished. In inerting a sealed area it is essential to ensure that the area is as tightly sealed off as possible, on one side at least, in order to keep the nitrogen requirement to a minimum. The outlet of an inert gas delivery line should be 20 m beyond a seal.

The more airtight the sealing, the more quickly the neutralization procedure will be completed. It can be seen from Figure 5-6 that to control a concealed fire ($O_2 < 1$ Vol. %) the leakage air flow over the source of the fire may be 3 m^3/min with an available 50 m^3/min of nitrogen; to control an open fire ($O_2 < 12$ Vol. %), the air current, with an available 50 m^3/min of introgen, must not exceed 45 m^3/min.

If the inert gas delivery system is interrupted during firefighting, rescue teams must be withdrawn.

During inerting the atmosphere on the return side may have increased carbon monoxide and methane levels and a reduced oxygen level. It may, therefore, be necessary to disconnect electrical apparatus on the return side of the fire and to prohibit travelling on the roadways affected. Remote sampling facilities are particularly valuable in this context.

When holes are being drilled for the delivery of nitrogen, water or nitrogen should be used in place of compressed air as the flushing medium. Holes should be checked for potential leakage due to nitrogen back pressure. Nitrogen lances should be securely fastened in place.

Air samples should be taken hourly while any explosion potential exists, preferably by remote monitoring systems. If the risk of explosion diminishes the sampling interval can be increased.

Measurements should be taken outbye the inerting site and records kept of air quantity, carbon monoxide, methane and, if possible, oxygen. Where possible, an alarm system should be introduced to monitor and alarm the conditions outbye the inerting site.

Local inerting has recently been used in addition to sealing off in fighting fires caused by spontaneous combustion in the goaf of an active mining area. In this case, work should only be carried out by suitably trained personnel under adequate supervision. Persons must be withdrawn from the workings if there is any risk from fire or combustion gases.

When a concealed fire is being combatted in the goaf of a mining area, hazards may arise from combustion gases or from the after effects of combustion of explosive gas mixtures. As a general rule, work should only be considered in the following circumstances:

- the position of the concealed fire is known or can be established with reasonable accuracy.
- the goaf is sealed off from the active workings by monolithic gateside packs and where necessary with additional sealing
- on the basis of measurements and monitoring of sampling pipes it has been established that the gas mixtures flowing from the seat of the fire are non-explosive
- the Graham ratio is lower than 1
- any increase in the quantities of carbon monoxide flowing from the suspected seat of the fire is low

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- in the open workings the maximum permissible noxious gas concentration in the ventilation air has not been exceeded and the oxygen content is greater than the minimum requirement.

If the CO generation rate is 20 L/min or more from an inaccessible fire, only rescue teams should carry out remedial work and normal production must be suspended.

It must be recognized that deep seated fires in a production cannot generally be extinguished by inerting though they may be controlled sufficiently to allow mining to proceed.

When coal is left in a mined-out area and leakage cannot be completely eliminated, pressure balancing and/or preventative inerting may be used. In these circumstances, the objective is to replace fresh air leakage with nitrogen.

1.3.5 Recovery After Inerting

In the past it has frequently been necessary to keep a sealed-off fire zone closed for months to cool and extinguish an exposed fire. For spontaneous combustion fires experience in some coalfields has shown that a sealed off period of at least 6 months is required. In such cases the workings often suffer from the effects of rock pressure and stagnant water and the work of firefighting teams has often been difficult since the temperature of stagnant water reaches the rock temperature and the relative humidity approaches 100%. Special measures have frequently been necessary when explosive gas mixtures were expected during the re-ventilation of the workings.

Inerting has removed the clear-cut distinction between the sealing-off and reopening of fire zones. There is now a more progressive transition from fire control to the resumption of normal mining operations. The rapid control of a fire by an inert atmosphere obviates many of the disadvantages mentioned above and provides an important saving in time. Inerting makes it possible to create in a fire zone gas mixtures with no explosion risk even when air is introduced. When monitoring shows that the fire has been extinguished and that explosive mixtures cannot be produced by admitting air the fire zone can be recovered by controlled reventilation.

A further benefit from using nitrogen is that considerably fewer negative effects such as strata pressure, water or roof falls, can be expected in the fire zone. Climatic improvements in the fire zone from the discharge of cold and dry nitrogen enable rescue teams to be in action for longer periods.

After reventilation subsequent work can be conducted in a normal atmosphere without breathing apparatus. However, workings should not be manned until the rescue team has established that all areas are free from noxious gases and that the fire has been extinguished beyond doubt.

5.3.6 Reporting

To enable the coal industry to benefit from inerting experience a report should be prepared for each incident. The report should include at least the following:

- name of mine
- date of occurrence
- reason for inerting
- start of inerting

- duration of inerting
- energy used
- delivery distance
- pipe diameter
- delivery pressure
- temperature of inert gas
- surface temperature
- maximum flow rate of nitrogen
- average flow rate of nitrogen
- total nitrogen consumed
- any technical problems
- result of inerting

The completion of such reports will facilitate exchange of information between operators and rescue services in Canada and overseas.

5.4 COSTS

The cost of using nitrogen to combat mine fires is made up of the cost of liquid nitrogen used, the operating cost of the evaporating unit and the labour and material costs for conducting the inerting operation in the mine.

There is no Canadian experience of fighting mine fires with nitrogen from which to quote actual costs but in Germany and the United Kingdom the average cost of gaseous nitrogen is between \$0.15 and \$0.25 per m^3.

The direct cost of fire control with nitrogen can be high. For example, the Daw Mill incident on 15's panel in 1980/81, cost more than the equivalent of \$0.92 million. However, equipment costing \$6.8 million was preserved and 0.9 Mt of coal reserves were saved.

In Germany during the period 1974 to 1979 a total of about \$25 million was expended on inerting and 30 sets of longwall equipment were saved.

In most mine fire situations the major financial loss results from the interruption of production while the fire is being tackled and in many cases a further period while the fire area remains sealed off to allow safe conditions to develop for re-entry. Where inerting is not used the time lapse between sealing off and reopining can often be many months, because even fires of limited size frequently set alight surrounding strata that remain hot even after the initial fire source has been removed. The principal economic advantage of inerting is that it can rapidly extinguish the fire and significantly reduce the extent of burning. Thus re-entry into sealed areas can be undertaken at an earlier date with consequent financial benefits.

Various commercial arrangements have been established overseas between nitrogen suppliers and mining companies including outright purchase of the evaporation unit and purchase of liquid nitrogen as required; evaporator rental with special 'on call' charge; and liquid nitrogen charges that decrease on a per m^3 basis according to consumption. Canadian mine operators should initiate discussions with nitrogen contractors to determine the terms under which nitrogen and evaporation equipment could be provided.

5.5 TRAINING

Training programs should be developed and implemented by mining companies to ensure that management and other personnel fully understand the principles and procedures of inerting. The detailed content of training programs will vary according to the equipment and systems adopted for a particular mine. Discussion in small work groups and the combat of 'mock' emergencies with inerting, have been found to be valuable training techniques. This training should complement normal firefighting and emergency training.

Courses should be run for:

- management and engineers
- rescue teams
- ventilation specialists
- supervision and workforce.

Management and engineering level courses should include the theory and practice of inerting, consideration of the mine emergency plan for inerting and should review inerting experience through case studies.

Rescue teams should be made familiar with the inerting plan for the mine and with the equipment and systems to be used in an emergency. They should be given hands-on training in making ready pipe ranges, purging and inerting in a variety of simulated fire situations.

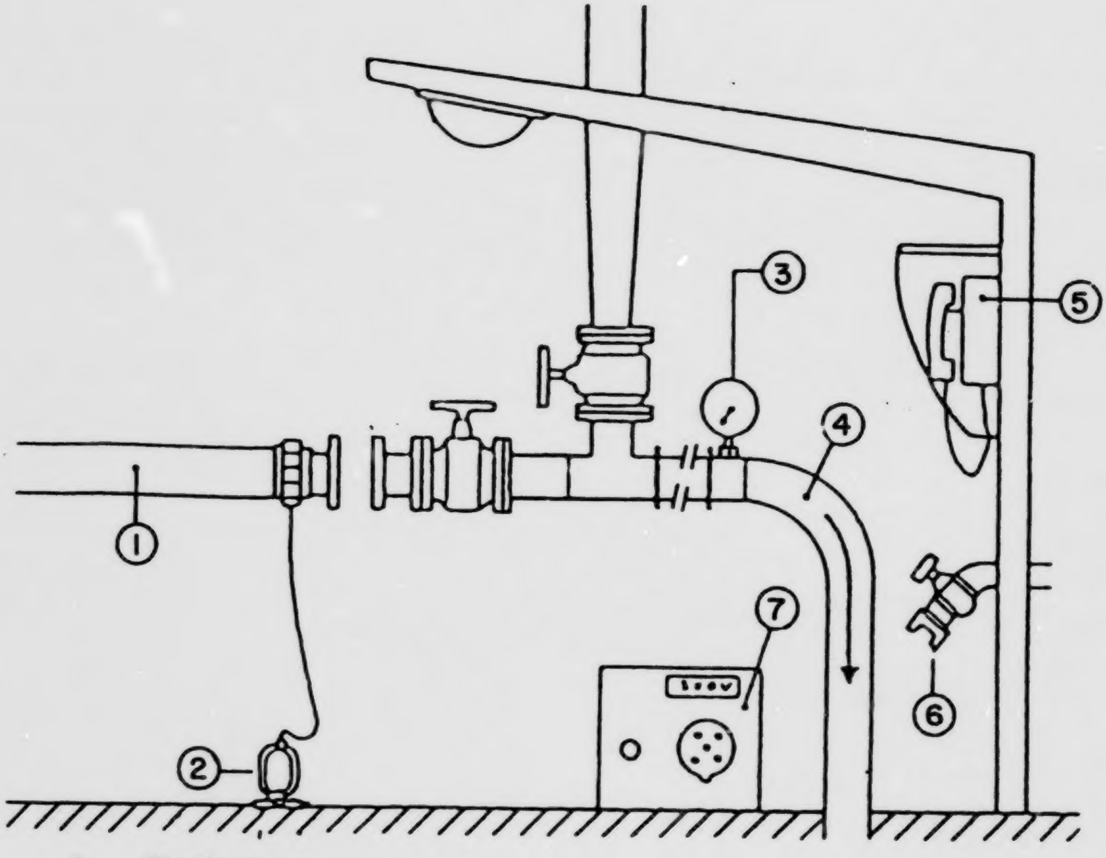
Ventilation specialists should be familiarized with systems and procedures for inerting with emphasis on monitoring and controlling the nitrogen flow and the mine atmosphere.

Mine supervision and workforce should be familiarized with inerting system and methods and should be trained in their specific roles in an inerting application.

Training should stress that inerting is one of many techniques that can be used to combat a mine fire and should be provided in the context of an overall application of mine fire control.

FIGURE 5-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SURFACE CONNECTION FOR NITROGEN



1. **Pipe from evaporator**
2. **Grounding**
3. **Pressure gauge**
4. **Pipe to mine**
5. **Phone**
6. **Water supply**
7. **Electrical supply**

FIGURE 5-2a

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FLOW OF NITROGEN

When the Pressure is 5 Atmospheres

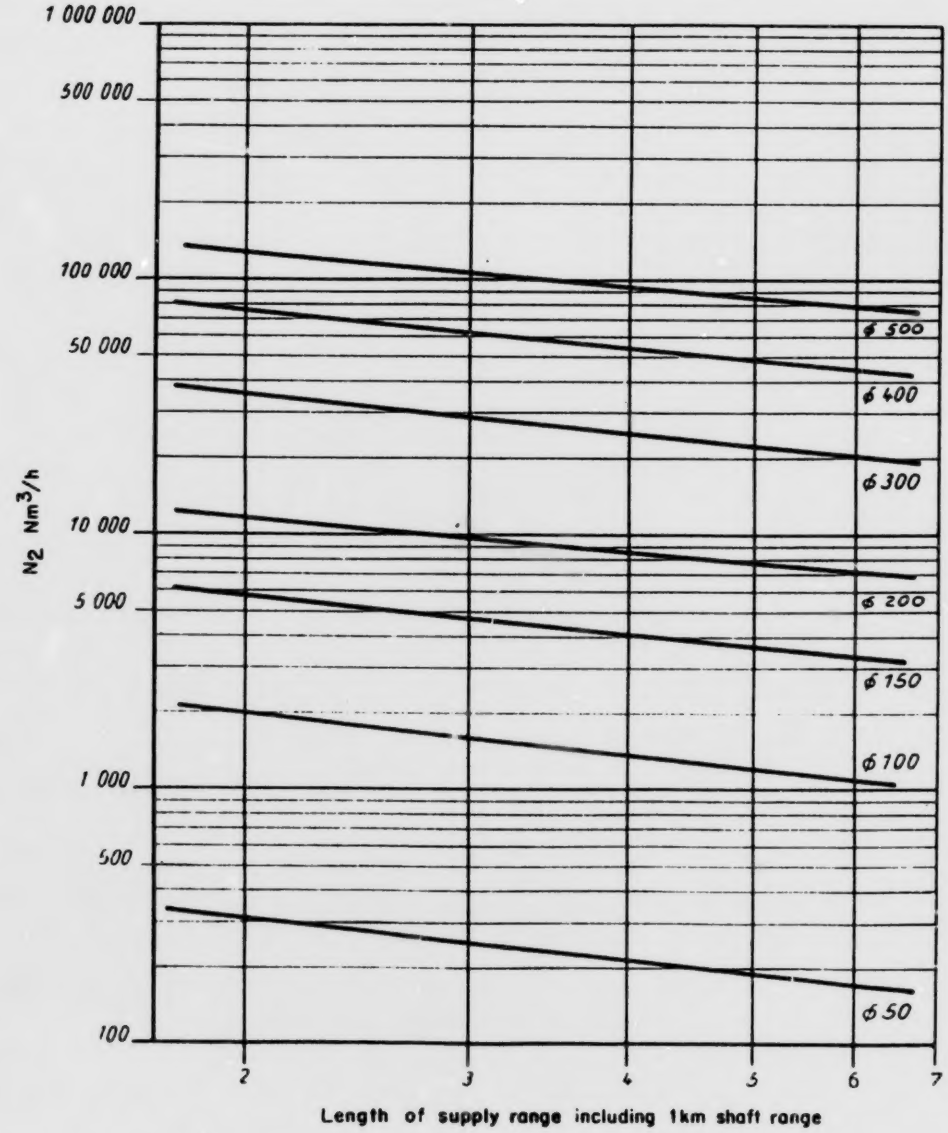


FIGURE 5-2b

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FLOW OF NITROGEN

When the Pressure is 10 Atmospheres

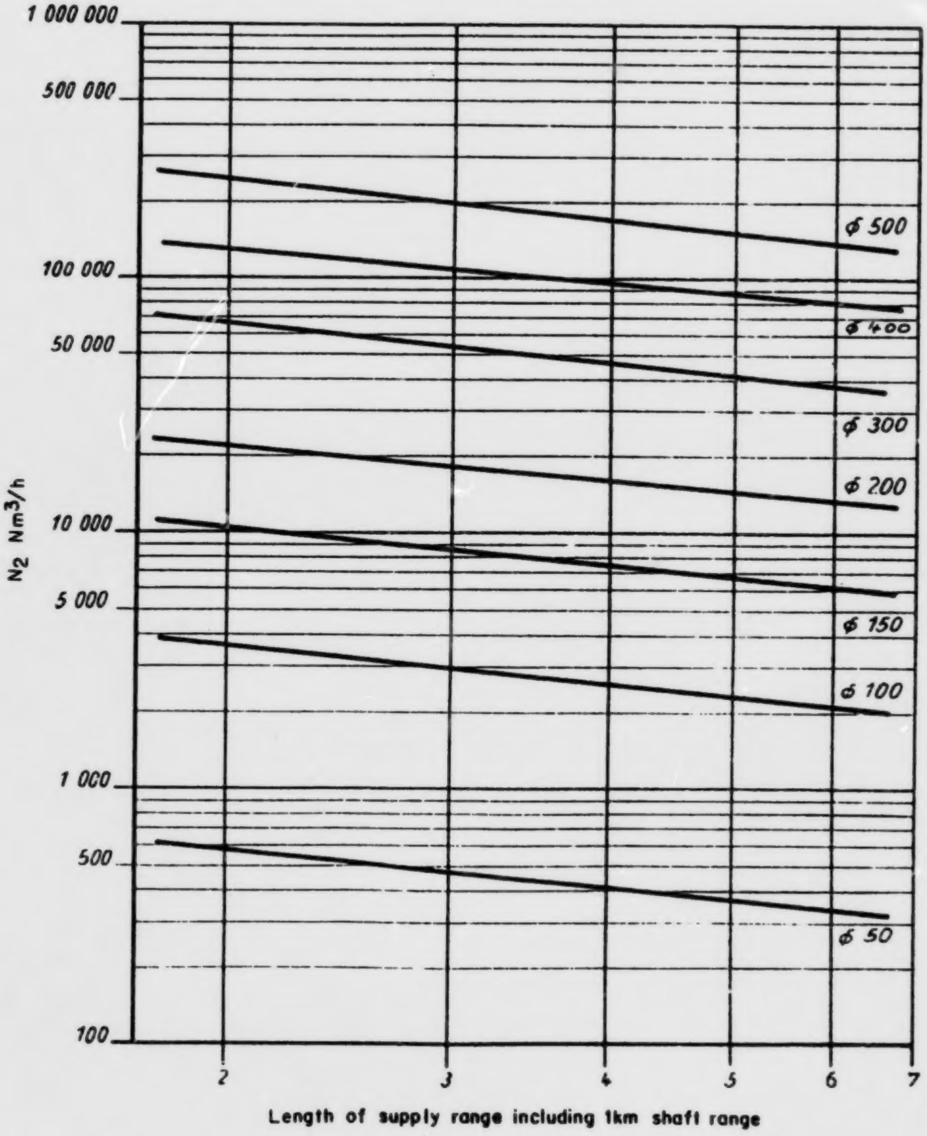


FIGURE 5-2c
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FLOW OF NITROGEN
When the Pressure is 15 Atmospheres

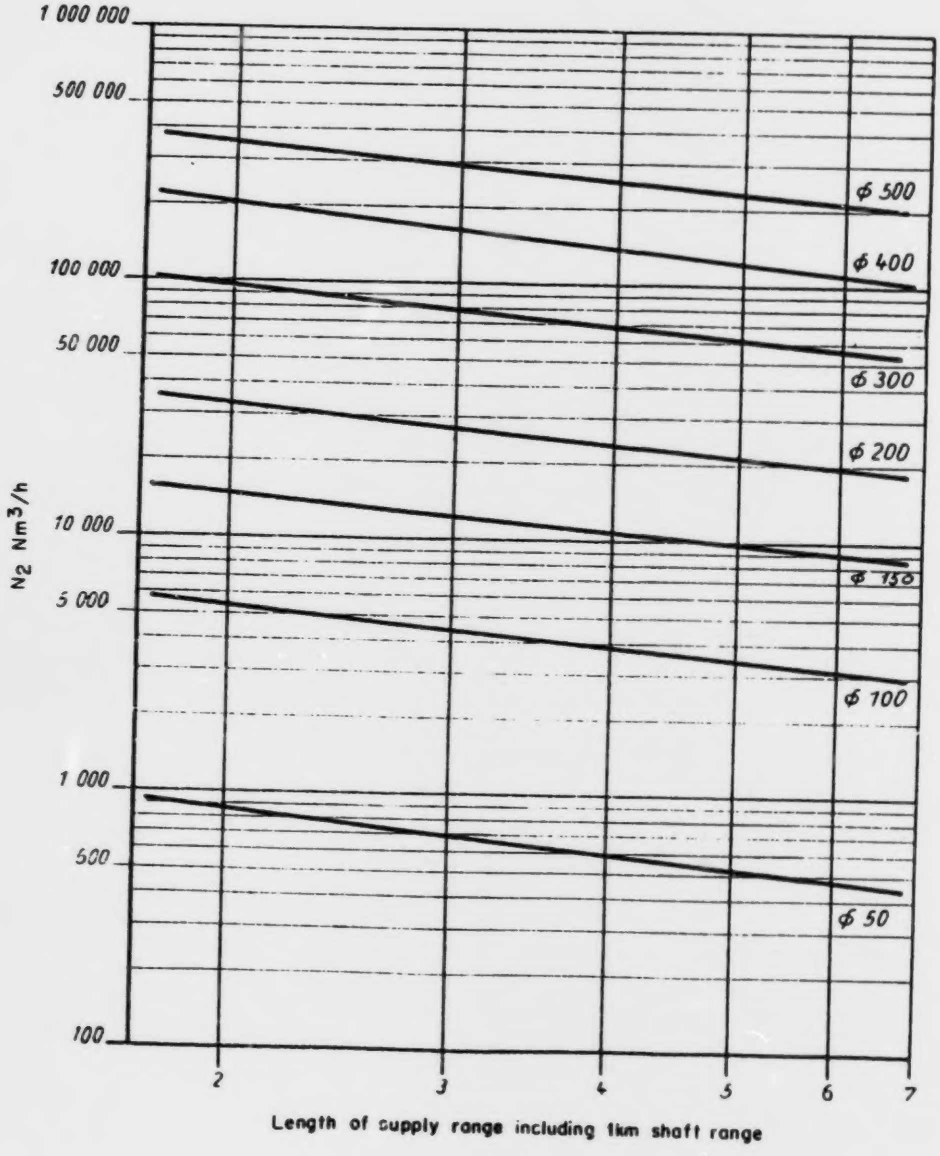


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FIGURE 5-3
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

MANIFOLD FOR NITROGEN DISTRIBUTION

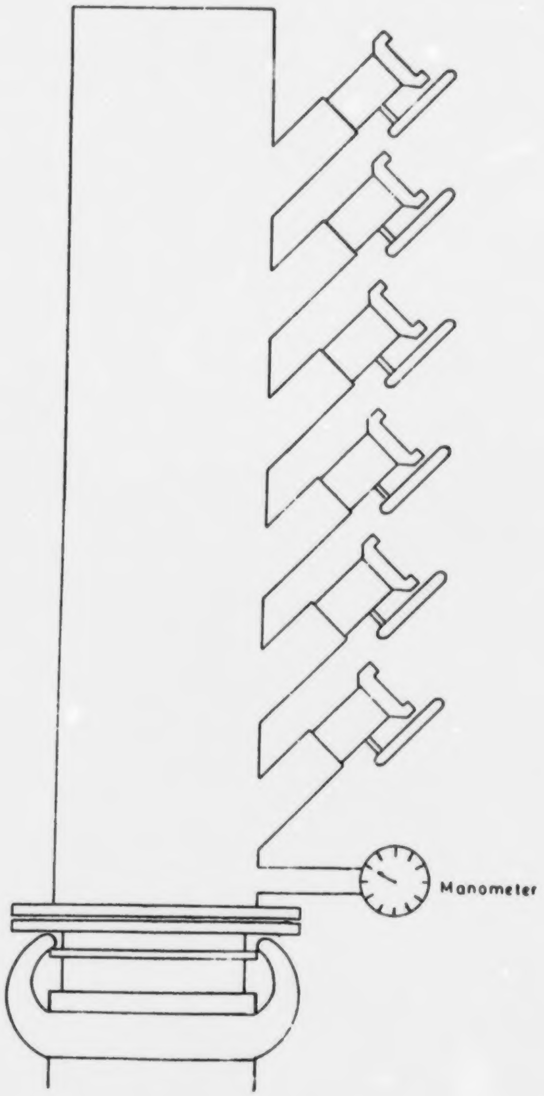


FIGURE 5-4

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

EXAMPLE OF SURFACE PLAN OF INERTING ARRANGEMENTS

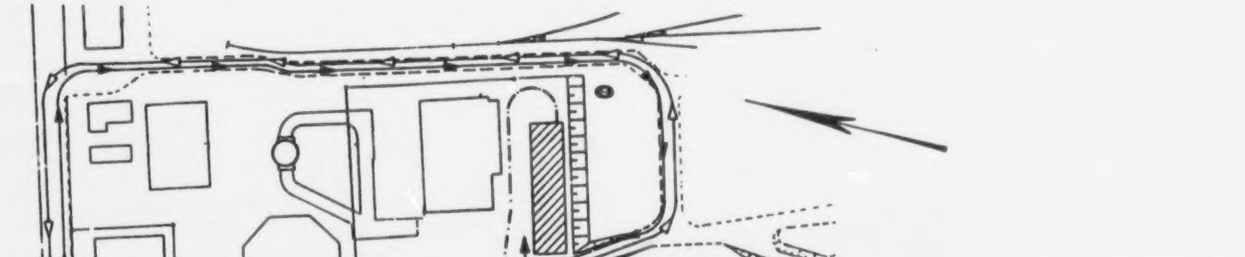


FIGURE 5-5

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

EXAMPLE OF UNDERGROUND PLAN OF INERTING ARRANGEMENTS

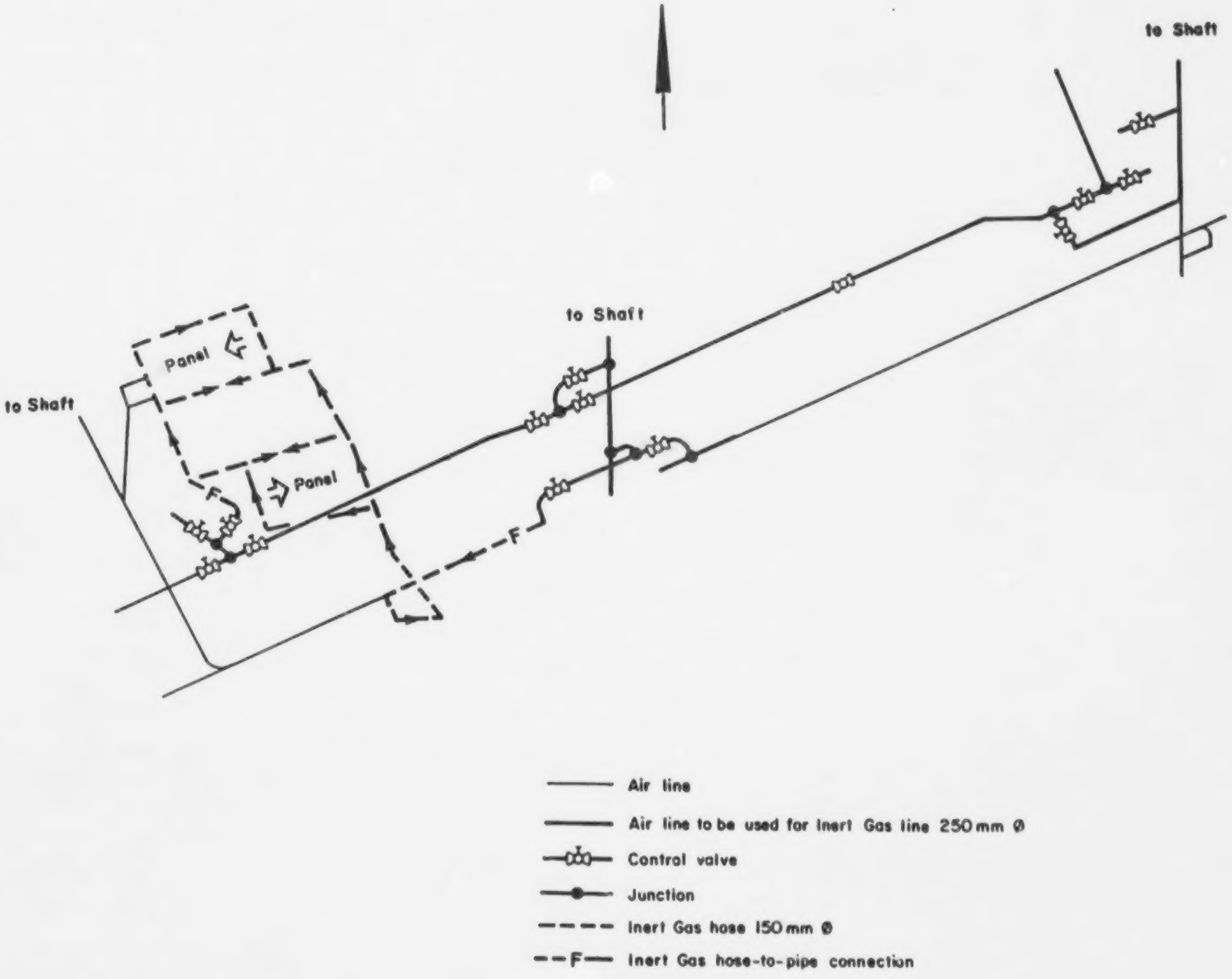
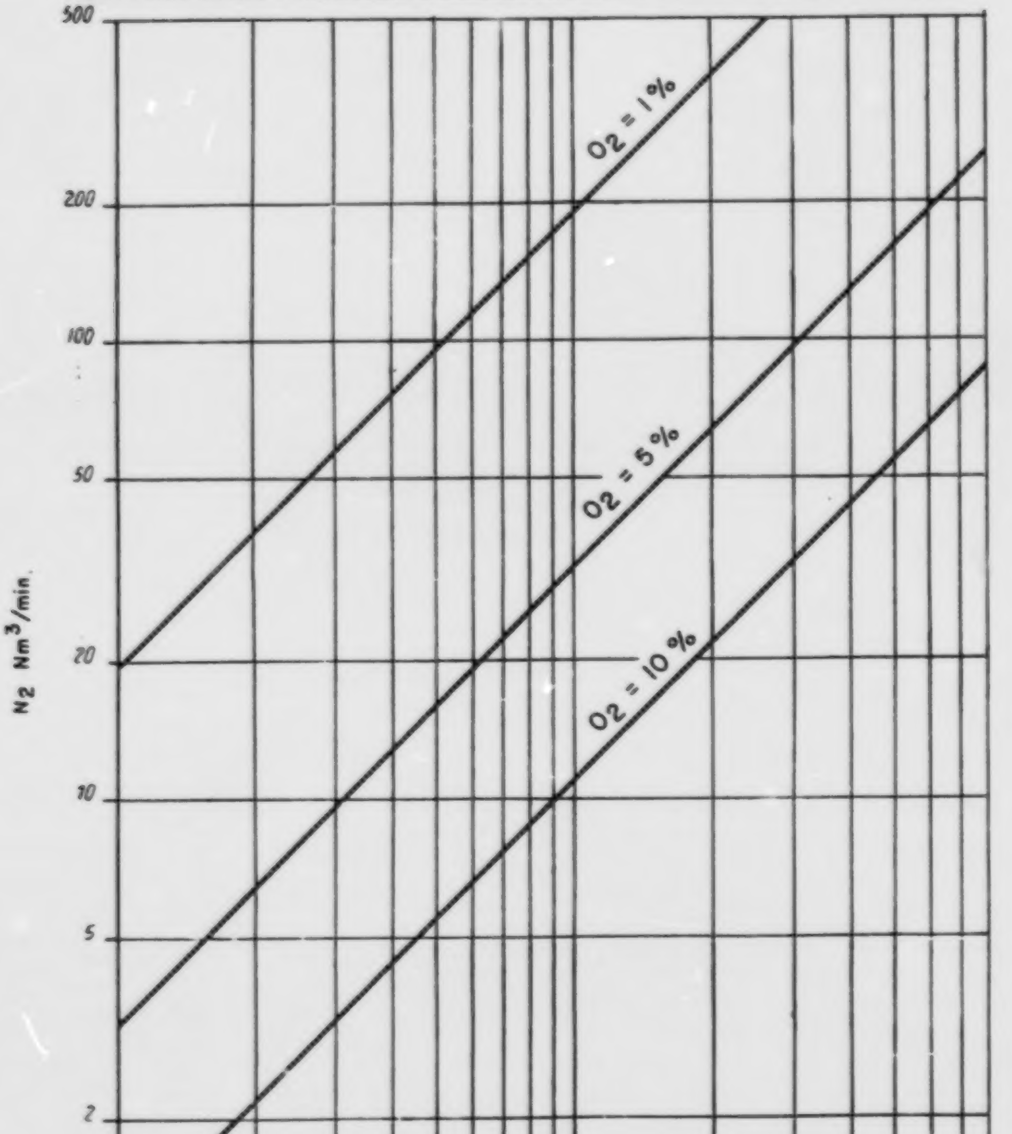


FIGURE 5-6

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

RELATIONSHIP BETWEEN AIRFLOW, NITROGEN AND OXYGEN



UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SECTION 6 - FUTURE DEVELOPMENTS

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SECTION 6 - FUTURE DEVELOPMENTS

6.1 GENERAL

Though control of mine fires by inerting is well-established world-wide, there are areas where modifications to mine design and further research and development could increase the benefits to be derived from the technique. In this section some mine design considerations are reviewed and certain promising activities are identified that are currently being carried out by research & development organizations offshore. Areas where canadian research & development could be beneficial are identified.

6.2 MINE DESIGN IMPLICATIONS

Most inerting has been carried out in response to emergency situations within mine designs that have not been influenced by the availability of inerting technology. An exception is in the field of spontaneous combustion control by long-term injection of nitrogen. In this context, the design at Rozelay Mine in France includes special nitrogen ranges which enable nitrogen to be fed systematiclly into inaccessible mined-out areas to control incipient heating. The integration of inerting into the mine design has enabled Rozelay Mine to operate the 'soutirage' system in seams susceptible to spontaneous combustion with a high degree of control.

A similar but more ad hoc approach has been adopted at mines in the UK South Midlands Coalfield, where thick seams prone to spontaneous combustion are mined by multi-slice longwall. Inerting is applied as needed, depending on the results of air analyses. In this coalfield plans for the withdrawal of longwall face supports include the introduction of nitrogen into the waste during recovery of equipment from worked-out panels. The degree of control provided by nitrogen injection enables equipment recovery and transfer to be more reliably scheduled.

In Westar Mining's hydraulic mine a number of fires have occurred as a result of spontaneous combustion. This hazard is recognized and mine design includes substantial pillars between mining blocks, principally to provide effective seals and to isolate possible fires. A considerable amount of coal reserves are tied up in these pillars. Consideration could be given to reducing the extent of such pillars if spontaneous combustion in the mined-out areas could be effectively controlled. Further investigations are required to evaluate the feasibility of using such an approach.

Spontaneous combustion is not a serious problem in other Canadian coal mines, but should underground mine development take place in spontaneous combustion prone areas, such as the Alberta Plains, then the implications of the availability of preventative inerting technology should be taken into account during the design of the mine design phase.

6.3 RESEARCH AND DEVELOPMENT

Current research and development work related to inerting is being conducted in several countries. Some of the more interesting projects include:

- **Small Portable IGG**

  Work is being carried out in Poland to develop a highly portable IGG for underground use.

- **Liquid Nitrogen Supply System**

  Consideration has been given in the Lorraine coalfield, France to connecting high risk mines to commercial liquid nitrogen plants so that liquid nitrogen can be piped directly from the manufacturer to the evaporating unit at the mine. A grid is envisaged to feed a number of mines in this coalfield.

- **Permanent Evaporation Plants**

  Consideration has been given in the UK to the establishment of permanent nitrogen production facilities on the surface of high risk mines. It is believed that the readily available supply of nitrogen at low cost will facilitate inerting to pre-empt potential spontaneous combustion hazards.

- **Nitrogen Cooled IGG**

  Development is continuing in Poland with nitrogen cooling of the GCP from IGG.

- **Goaf Air Flows**

  The need to develop more understanding of air flows in mined-out areas has been identified both in Canada and in the UK. Some work has already been undertaken in this field but further efforts are needed. The results will assist in planning and implementation of the inerting of goaf areas.

- **Surface Sited IGG**

  In Germany and Poland the need has been identified for a high capacity IGG that can be sited on a mine surface and whose GCP can be transmitted long distances in small diameter pipes. The successful development of such a machine could significantly reduce the cost of inerting. Problems to be overcome include the large amount of water vapour and the high temperature of the products leaving the generator.

- **Air Separation Units**

Russian literature refers to the existence of a minesite air seperation unit capable of 100 m^3/min nitrogen flow. This is significantly higher than mobile air separation units operating in the West. In view of the cost advantage of nitrogen produced by air separation compared with that evaporated from liquid nitrogen, work should be undertaken in the West to investigate a higher capacity ASU.

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UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

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7.8 U.S.A.

Case Study #17 - Inland #1 Mine 7-104
Case Study #18 - Joanne Mine 7-109
Case Study #19 - Olga Mine 7-115
Case Study #20 - Peabody No. 10 Mine 7-122
Case Study #21 - River King Mine #1 7-129

7.9 U.S.S.R.

Case Study #22 - Conarnaya Mine 7-134

CANADA

#26 COLLIERY

TYPE OF MINING	Longwall
DATE OF INCIDENT	1975
TYPE OF FIRE	Open fire on advancing longwall
INERTING AGENT USED	Carbon Dioxide
DELIVERY SYSTEM	Liquid carbon dioxide vaporized on surface and delivered by pipeline to face
GAS DELIVERY RATE	-
TOTAL QUANTITY INERT GAS USED	125 000 m^3 CO_2
SPECIAL FEATURES	Use of carbon dioxide as inerting agent and only experience of using inert gases to control mine fires in Canada.
REFERENCE	Leblanc, W. and Phalen, J. 1976: Fire in No. 11 South, No.26 Colliery, Glace Bay; CIM Bulletin, v. 69/774, (Oct), p. 117-124.

1. BACKGROUND

11S panel of the 2.2 m thick Harbour Seam in #26 Colliery, Glace Bay was worked by a 183 m long advancing longwall face, equipped with a double ended ranging drum shearer and chock supports (Figure 1-1). The Harbour Seam is gassy and characterized by the occurrence of sandstone intrusions. Measures were taken to prevent frictional ignitions. A methane drainage system extracted 850 m^3/h methane from holes drilled over the gob area.

On June 19, 1975 an ignition occurred at the shearer causing a fire that travelled some 30 m along the face and set the coal alight. Attempts to fight the fire with CO_2 fire extinguishers, stonedust and water failed after a fall exposed a methane drainage hole, allowing the gas to be ignited.

The decision was made to seal off the fire with stoppings at the 11S coal road and in the airway below 9S district. Access to a site closer to the fire on the return side was impossible due to smoke and heat. Water was also fed into both the 11S road and materials road, which because of the seam geometry, provided an effective seal on the coal road. Doors and stoppings in the area were sprayed with Mandoseal to reduce air leakage to the fire.

On June 21, 1975 air analyses behind the seal at 9S indicated an explosive mixture as follows:

$$3.6\%\ CO_2 : 6.96\%\ CH_4 : 14.90\%\ O_2 : 0.20\%\ CO$$

In order to control the fire it was essential to be able to erect a seal on the 11S material road inbye of the return airway. It was decided to pump CO_2 into the district to provide an inert atmosphere in which the rescue teams could operate and which would extinguish the fire.

2. CARBON DIOXIDE SUPPLY AND DISTRIBUTION

Liquid Carbonic Canada Ltd. of Montreal supplied liquid CO_2 by rail in 80 t capacity tankers. The CO_2 at 350 psi was passed through an evaporator heated by steam from the colliery boilers. The CO_2 gas was then piped to 11S face via the existing water line.

3. OPERATIONAL PROCEDURE

232 tonnes of CO_2 gas were pumped on to 11S face between July 3 and July 12, 1975. On July 8, 1975 rescue teams were able to enter the material road and erect seals inbye of the return airway. Air analyses demonstrated the effects of the procedure:

	O_2%	CH_4%	CO%	CO_2%	H_2%	N_2%	C_2H_6%
July 8 Prior to seals	10.6	12.6	0.60	14.1	0.06	63.0	-
July 9 After seals Erected	2.7	23.0	0.80	26.6	0.08	46.0	-
July 14	0.80	55.0	0.10	22.7	-	18.2	2.7
October 3	0.90	69.9	0.004	3.2	-	21.5	3.6

The booster fans for the mine were restarted on July 14 and mining operations commenced on 11N face on July 22.

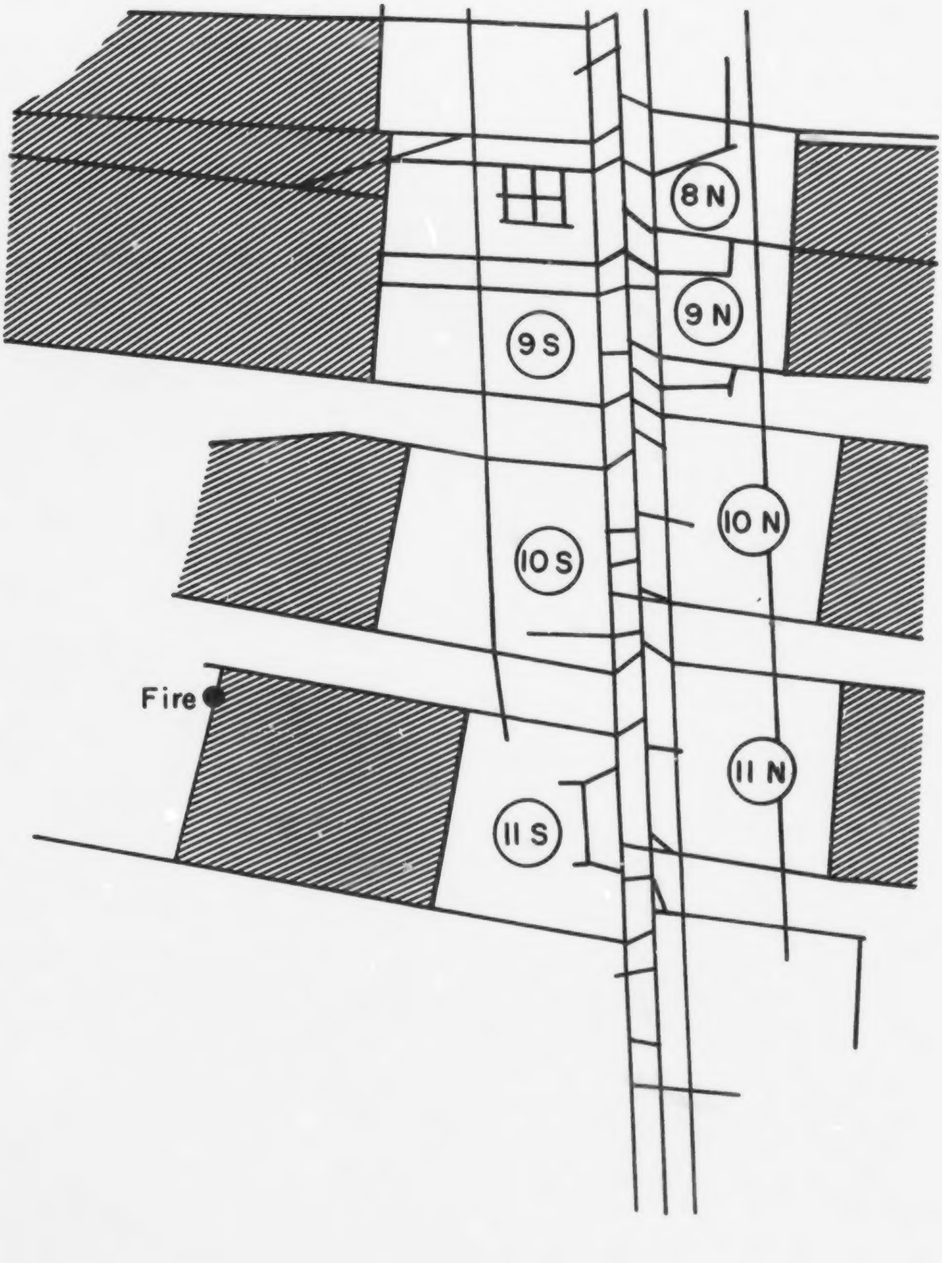
The district was re-entered on November 5, 1975 after the water seal had been lowered and air allowed in. Extensive fire damage was evident, and equipment was a total loss except for 60 m of face equipment and coal road equipment. By November 7, 1975 however, a temperature rise was apparent in the 11S material road so the district was permanently sealed off.

4. COMMENTS

This example demonstrates the use of carbon dioxide during the combat of an open fire on an advancing longwall face and the production of an inert atmosphere that allowed rescue teams to enter the area to erect seals.

FIGURE 1-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

PART PLAN OF THE ARCH DEEPS
#26 COLLIERY



CZECHOSLOVAKIA

DOUBRAWA MINE

TYPE OF MINING	Longwall
DATE OF INCIDENT	1978
TYPE OF FIRE	Methane ignited in heading
INERTING AGENT USED	Gaseous combustion products
DELIVERY SYSTEM	GAG inert gas generator
GAS DELIVERY RATE	Average 1000 m^3/min
TOTAL QUANTITY INERT GAS USED	490 000 m^3
SPECIAL FEATURES	
REFERENCE	Kukuczka, A. - personal communication: 1984: **Doswiadozenia zakcji w kopalni Doubrawa**, (Fighting a Fire at the Doubrawa Mine); Czechoslovakia.

1. BACKGROUND

The Doubrawa Mine, located in the Ostrava-Karvina region of Czechosolovakia, works the 5.5 m thick #3 seam by a longwall retreat method. The gassy seam dips at 20°.

A methane ignition on February 13, 1978 caused a fire in the gateroad at the face end. Because of the risk of a methane explosion, it was decided to utilize the GAG inert gas generator, brought in and operated by the Polish Mine Rescue Service, to allow construction of a seal in a non-explosive atmosphere.

2. INERT GAS SUPPLY AND DISTRIBUTION

The GAG inert gas generator was installed underground, discharging cooled gases through a fire stopping.

3. OPERATIONAL PROCEDURE

The mine layout of the fire zone is shown in Figure 2-1.

The GAG arrived at the mine late on February 13, 1978. There were problems installing the machine in the 20° incline of the drift and with removal of the hot cooling (70°C) water from the machine. Fuel supply was also a problem since the GAG blocked the operation of the monorail transport system. These problems were resolved and the GAG was started at 10:10 pm on February 14, pushing inert gases through a gypsum stopping T_1 by 10:45 pm. However, during the operation the GAG's engine slowed down and its temperature increased, which was thought to be caused by a rock fall in the fire zone. The GAG was run for 8-1/4 hours after which the composition of the gases in the return airway were oxygen 5%, carbon dioxide 2% and carbon monoxide 10.5%

and the temperature was 55°C. The total volume of inert gas fed into the fire area was 490 Mm^3.

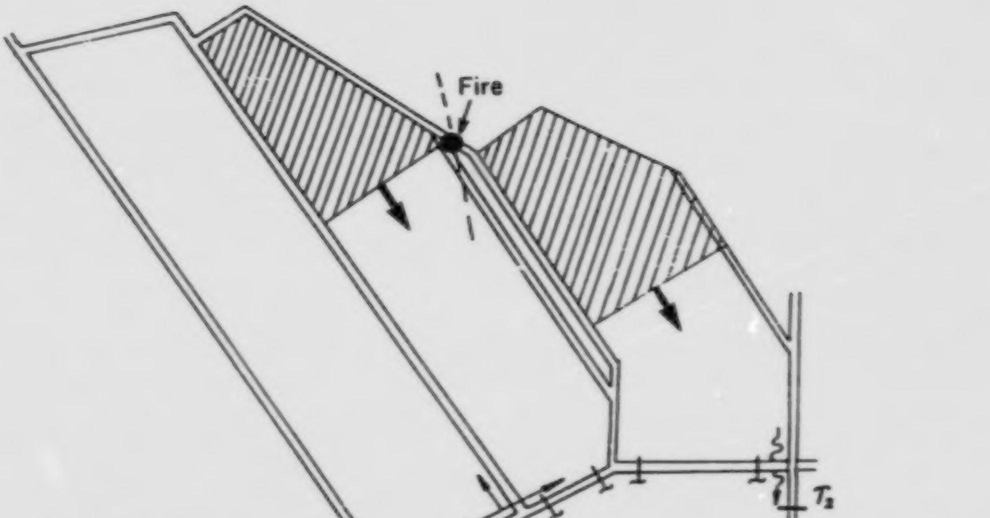
To seal the area, a stopping was required in the return airway. The site for the stopping was filled with smoke. It was found that a compressed air line behind the inlet stopping was open. When it was closed, conditions improved to allow the return stopping to be constructed and closed by February 16 at 4:30 pm.

4. COMMENTS

This case study demonstrates how the use of the GAG inert gas generator allowed a fire area to be sealed safely by reducing the risk of a methane explosion.

FIGURE 2-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

DOUBRAWA MINE LAYOUT



CZECHOSLOVAKIA

FUCIK MINE

TYPE OF MINING	Longwall
DATE OF INCIDENT	1977
TYPE OF FIRE	Methane ignition in heading
INERTING AGENT USED	Gaseous combustion products
DELIVERY SYSTEM	GAG inert gas generator located underground
GAS DELIVERY RATE	900 m^3/min
TOTAL QUANTITY INERT GAS USED	390 000 m^3
SPECIAL FEATURES	Use of GAG to reduce risk of methane explosion during sealing of a fire area
REFERENCE	Kukuczka, A. - personal communication: **Doswiadozenia w akcji w kopalni Fucik,** (Control of a Fire at the Fucik Mine); Czechoslovakia.

1. BACKGROUND

The Fucik Mine is located in the Ostrava-Karvina region of Czechoslovakia. On November 18, 1977 methane was ignited by sparks during withdrawal of steel supports from a gateroad of a retreat longwall face (Figure 3-1). The fire spread to the coal and was impossible to extinguish by direct attack methods. It was decided to seal the area, but because of the high methane emissions and high temperature (80°C) this was considered unsafe without the use of inerting techniques. A decision was, therefore, taken to use the GAG inert gas generator.

2. INERT GAS SUPPLY AND DISTRIBUTION

A GAG inert gas generator was supplied from Poland to be installed underground, discharging GCP through a fire stopping.

3. OPERATIONAL PROCEDURE

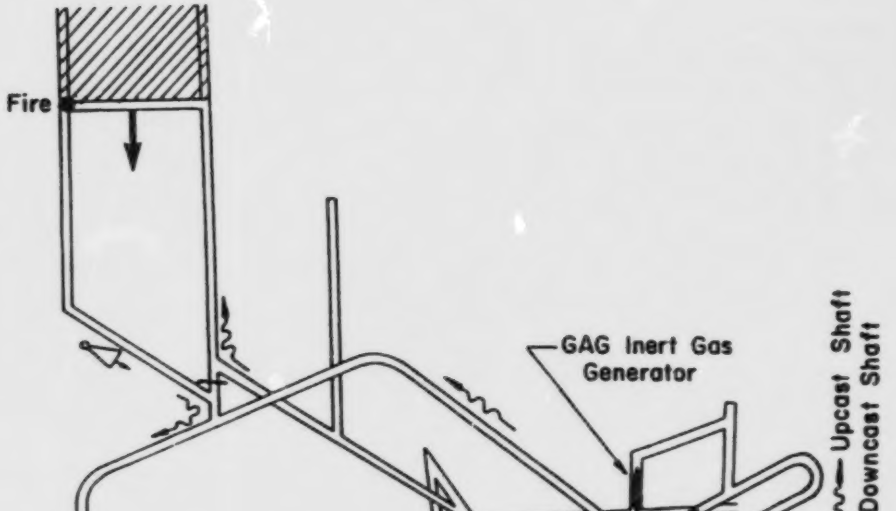
The GAG was installed in an intake roadway by November 19 (Figure 3-1). After three hours operation the bulk of the oxygen had been removed from the 50 000 m^3 fire area and the temperature lowered to 54°C. The GAG was operated for a further 3-1/2 hours when the oxygen content of air at the site of the return stopping was 5% at 46°C. The return stopping was constructed and the area sealed off. The mine was reopened on November 21. After two weeks it was possible to locate stoppings closer to the face and two weeks later the face was reopened.

4. COMMENTS

This case study demonstrates how the GAG inert gas generator was used to reduce the risk of explosion during the sealing of a fire area and to lower the temperature of the area.

FIGURE 3-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FUCIK MINE LAYOUT



Case Study #4

FRANCE

L'HERAULT MINE

TYPE OF MINING	Development decline of uranium mine
DATE OF INCIDENT	N/A
TYPE OF FIRE	Open fire on conveyor belt
INERTING AGENT USED	Carbon dioxide
DELIVERY SYSTEM	Liquid carbon dioxide injected through portal stopping
GAS DELIVERY RATE	66 m^3/min (approximately)
TOTAL QUANTITY INERT GAS USED	10 000 m^3
SPECIAL FEATURES	Use of carbon dioxide to extinguish open fire in development decline of uranium mine
REFERENCE	Marvy, A. 1981: **Un Procede Original de Lutte Contre l"Incedie en Travaux Miniers Souterriams;** (An Original Method of Fighting an Underground Fire), Industrie Mineralles Techniques, (Nov).

1. BACKGROUND

This case study describes a fire during the development of the L'Herault underground uranium property in the Mas Lavayre in France. At the time of the incident, two declines were being driven. The one in which the fire took place had a gradient of approximately 14° with a cross section 2.5 m high by 4 m wide. The drift, which had advanced some 700 m, was not connected to any other workings and was equipped with a belt conveyor whose drive head was located on the surface.

The incident occurred at 5 am at the start of the first shift of the day. At 5:15 am a workman noticed that the drive head was glowing and that the slipping driving drum had started a fire on the conveyor belting. The workman stopped the motors and having tried to deal with the fire using extinguishers without success, raised the alarm. At this moment, the belting broke and the top belt ran down into the underground workings. A warning was given immediately by telephone to the men underground at 5:15 am. Senior staff arrived on the site by 5:30 am and an appreciation of the situation was made.

All the workmen on the shift made their way back to the surface having attempted to put the fire out using extinguishers from along the decline. The fire area was about 70 m from the surface (Figure 4-1). The belt was piled up in an accordion fashion, resting on the structure, leaving space alongside for the workmen to escape. The auxiliary ventilation had been stopped.

Black smoke was coming out of the top of the portal and fresh air was entering at floor level. The fire appeared to be getting worse. It was considered critical to save as much as possible of the conveyor system and to re-establish pumping as soon as practicable to prevent damage to the motors on the equipment at the bottom of the drift.

A stopping was put at the entrance to the decline to reduce the inflow of fresh air. Pipes were placed in the bottom and top. After the stopping was completed the smoke and fumes were seen to reduce, with the fire under control but not extinguished. A means of completely extinguishing the fire without negative effects and without risk to human life had to be found. Sealing off was not thought to be desirable because of the time that would have been taken and the effect on the drivage and equipment.

The injection of inert gas would eliminate oxygen and purge the atmosphere in the decline so that recovery operations could be carried out, and would have a beneficial effect on the temperature. It was decided to use carbon dioxide because it was available in large quantities, could be transported in a liquid form, and by virtue of its density would progressively fill up from the bottom of the 14° decline to the surface, creating an atmosphere high in CO_2 and thus inert the fire. The low temperature of the gas would assist cooling.

2. CARBON DIOXIDE SUPPLY AND DISTRIBUTION

Carboxyque Francais supplied a 30-ton truck of liquid CO_2 by 8 pm. The liquid CO_2 was pumped through the stopping into the decline.

3. OPERATIONAL PROCEDURE

The CO_2 was pumped into the decline from 9:00 pm until 5:30 am when the truck was empty, at an average rate of 66 m^3/min. The CO concentration at the stopping reduced from several hundred ppm initially to almost zero at the end of injection. CO_2 concentrations at the stopping did not increase for a considerable time, indicating it was filling the decline from the bottom.

The next morning, a reconnaissance by two rescue teams from the Cevennes Coalfield in the vicinity of the stopping indicated an absence of oxygen in the decline, that the fumes had been cleared, a reasonable ambient temperature and a high concentration of CO_2. No dangerous concentrations of CO were discovered. A second examination found the conveyor belt still red for a length of 6 m to 10 m. It was extinguished using water hoses. At 10 pm the ventilation was put back into action to supply fresh air and clear the CO_2 out of the drivage. The next morning an investigation by the rescue team conducted throughout the length of the 700 m gallery confirmed that the atmosphere was respirable. Pumping began and the incident was closed 30 hours after its commencement.

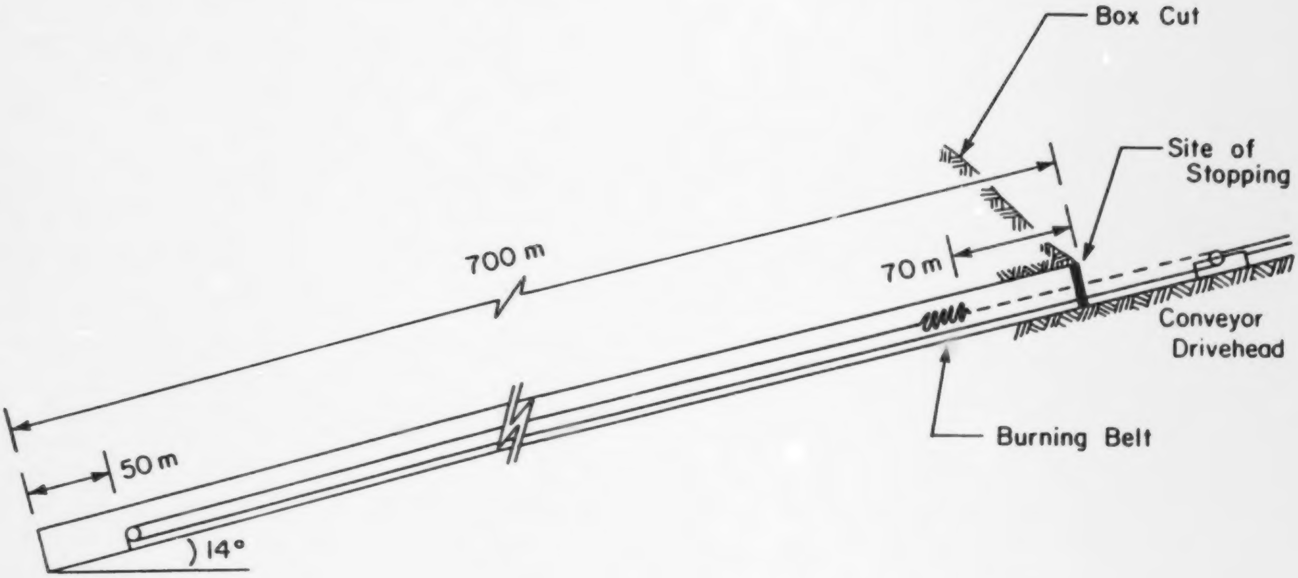
4. COMMENTS

The carbon dioxide had no negative effects on the drivage or its equipment. Risks to the rescue personnel were reduced and the cost of the stoppage of the drivage was minimized by the rapid control of the fire by this inerting technique.

This is an application of CO_2 in circumstances where its high density was considered beneficial. CO_2 gravitated to the bottom of the workings and backed up to the surface, driving out the post-fire atmosphere and cooling the workings.

FIGURE 4-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SCHEMATIC SECTION OF DRIFT AT L'HERAULT MINE



FRANCE

SAINTE FONTAINE MINE

TYPE OF MINING	Longwall
DATE OF INCIDENT	1980
TYPE OF FIRE	Ignition in zone of an overthrust fault along the face line of a retreating longwall
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Liquid nitrogen vaporized on surface and delivered by two pipelines to face
GAS DELIVERY RATE	Up to 210 m^3/min
TOTAL QUANTITY INERT GAS USED	3 000 000 m^3
SPECIAL FEATURES	
REFERENCE	Houilleres du Bassin de Lorraine 1980: **Cas de Lutte Contre un Incendie par Inertisation a l'azote dans un siege du bassin de Lorraine;** (Control of a Mine Fire by Nitrogen Inertization at a Mine in Lorraine Coalfield), Industrie Minerale les Techniques, #5-82, (May), p. 305-308.

1. BACKGROUND

A panel of 2.57 m working thickness in the F Seam of the Sainte Fontaine Colliery was worked by a 240 m long retreating longwall face equipment with a double-ended ranging shearer and hydraulic supports. The coal is high volatile bituminous and very gassy. The CO content is continuously monitored in the return air of the face with air samples taken in the tailgate and in the caved area.

On February 20, 1980, blue smoke was observed in the return air and the CO content rose from its normal level of 5 ppm up to 100 ppm. Flames appeared in the zone of an overthrust fault along the face (Figures 5-1 - 5-3). At the outlet of the surface fan the CO content rose to 1200 ppm, at which time the decision was taken to fight the fire by nitrogen injection in the maingate of the working.

2. NITROGEN SUPPLY AND DISTRIBUTION

Air Liquide delivered equipment within 10 hours of the request, comprising a mobile evaporator unit capable of delivering 240 m^3/h of nitrogen at a pressure of 15 bar. The nitrogen was delivered in liquid form in 26 m^3 tankers. Two further generators, having capacities of 100 m^3/min and 67 m^3/min were supplied from the Ruhr in Germany by Messer Griesheim Gmbh. These were heated by propane. The gas was piped to the face through two existing water pipelines, 3500 m long.

3. OPERATIONAL PROCEDURE

On February 21 a plaster stopping was commenced in the maingate. A rock fall, caused by the fire, reduced the airflow rate in the return from 17 m^3/s to 5 m^3/s. When the stopping was well advanced the air rate reduced to 2 m^3/s and nitrogen injection commenced at a rate of 128 m^3/min. On February 22, the second set of nitrogen evaporators were available and connected, raising the total injection rate to 211 m^3/min. Readings in the tailgate showed 800 ppm CO and 2% methane, and a temperature of 37°C. The stopping was sealed by the end of the day and the CO content dropped to 304 ppm. The next day, work commenced erecting a stopping in the returngate and was completed by February 24. During this period the CO content dropped progressively until it reached 4 ppm at the time the stopping was sealed.

The German evaporators were stopped on February 29 and the rate from the Air Liquide equipment reduced. On March 26 the use of nitrogen was discontinued.

A total of 3 000 000 m^3 of nitrogen gas was used at a cost of 2 million Francs.

4. COMMENTS

This example shows that by injecting nitrogen at the earliest opportunity a fire can be contained and brought under control rapidly, even though substantial quantities of nitrogen had to be used. A rock fall aided the control by reducing the air flow across the face.

FIGURE 5-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SAINTE FONTAINE MINE - GENERAL LAYOUT

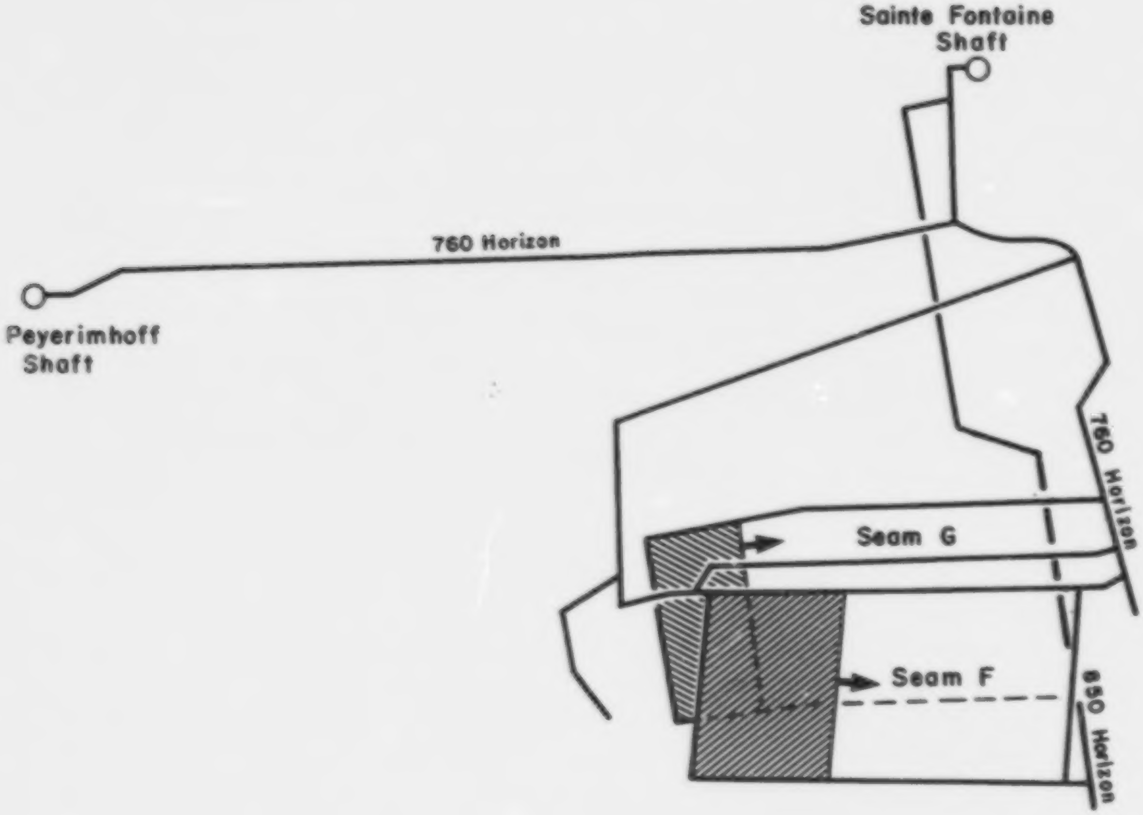


FIGURE 5-2
UNDERGROUND COAL MINE FIRE CONTROL
WITH INERTING SYSTEMS

PLAN OF DISTRICT - SEAMS F AND G

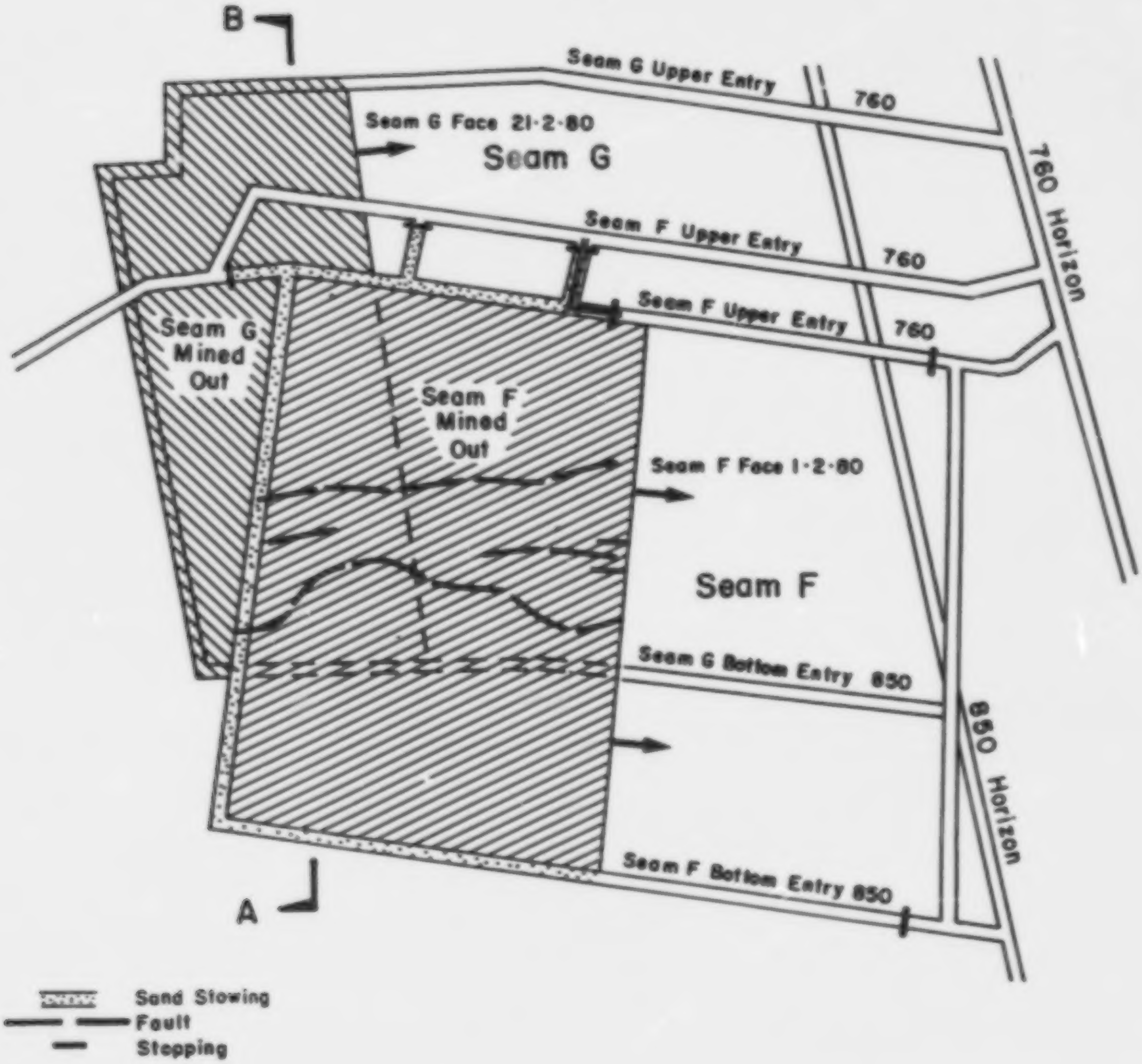
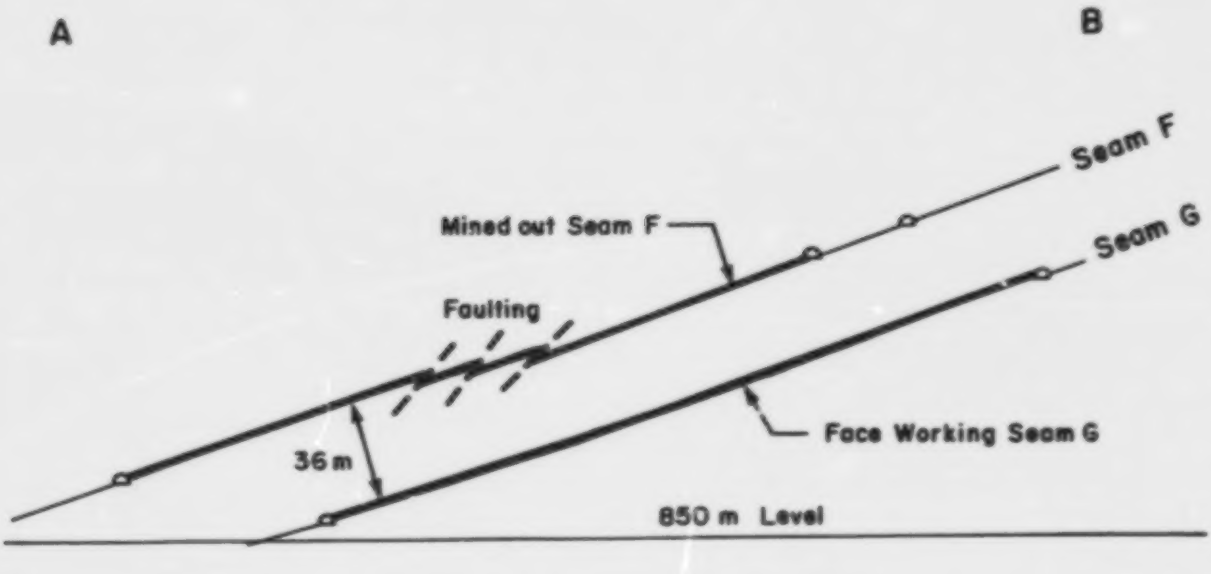


FIGURE 5-3
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SCHEMATIC CROSS-SECTION - 'AB'



FRANCE

ROZELAY MINE

TYPE OF MINING	Longwall with secondary caving ('soutirage')
DATE OF INCIDENT	1983
TYPE OF FIRE	Heating in goaf of retreat longwall
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Nitrogen from surface supplied into goaf through pipes which were extended as the face advanced
GAS DELIVERY RATE	Up to 25 m^3/min
TOTAL QUANTITY INERT GAS USED	
SPECIAL FEATURES	Use of nitrogen to control heating in goaf of active longwall and attempt to control fire in roof of entry
REFERENCE	Internal report, HBCM, France, November 1983

1. BACKGROUND

Rozelay Mine is located in the Blanzy Coalfield in central France. This shaft mine works the 'souterage' method, extracting thick seams of variable height by longwall retreat methods. The case study relates to face 66, a 100 m retreating panel extracting a height of about 2.75 m, followed by sub-level caving of the remainder of the slice. The seam dips moderately. At the time of the incident, face 66 had retreated more than 500 m. Roadways were supported by steel arches and face support was by special sub-level caving shields. Coal was mined by a double-ended ranging drum shearer.

The face was equipped with a nitrogen supply system, comprising a pipe range from the surface to the face that supplied nitrogen in controllable quantities to pipes in the goaf at the bottom end of the wall (Figure 6-1). New ranges were laid each 50 m of advance. Sampling points were maintained in the waste at the bottom, middle and top end of the face. The ends of the face opposite the top and bottom roadways were regularly sealed with 'Igloneige' foam to minimize air ingress into the waste. A remote monitoring system using Unor CO detectors was installed. In addition, regular samples were taken from the waste sampling points.

The first incident on face 66 was in March 1983. On March 18, sample point 91, 50 m behind the face (Figure 6-1), registered 7 ppm CO and 11% oxygen. 1.3 M^3/min of nitrogen were being injected through the pipe at point 93.

2. FIRST INCIDENT

The CO reading at point 91 rose from 260 ppm on the morning of March 22 and by 7 pm on that date had reached 12 600 ppm. The main return

Unor readings rose from 1 ppm in the morning to 43 ppm at midnight of the 22nd.

Action to counteract the heating included the injection of nitrogen behind the face at rates increased from 1.3 m^3/min to 2 m^3/min on March 24, and to 25 m^3/min on March 25. More than 175 000 m^3 were injected between 9 pm on March 23 and 9 pm on March 29. Tracer gas SF_6 was mixed with the nitrogen to assist the location of the path of the inert gas behind the face.

On March 22 the sub-level caving was discontinued. Face ventilation was reduced from 11 m^3/s to 6 m^3/s by short-circuiting some of the intake air. Additional 'Igloneige' foam sealant was applied at both ends of the face using 10 injection holes at each road head. Additional manpower was assigned to the face and six shifts were worked over the weekend to increase the rate of face advance. A hole was bored (Figure 6-1) to intersect the waste behind the face and injection of nitrogen commenced at 5:30 pm on March 24 at a rate of 8.3 m^3/min. Certain locations in the old workings behind the face were treated with foam to reduce potential leakage. The water seals into adjacent old workings were topped up.

The effect of these actions was to reduce to CO level from midnight March 22 (Figure 6-3). By March 26 at 1:00 am, the main return CO value had reduced to zero ppm.

As of April 21, 9.2 m^3/h of nitrogen were being injected behind the face. Face ventilation was 8.3 m^3/s and the readings from point 91 were zero ppm CO and 4.5% oxygen.

It is believed that the principal cause of the heating was a result of air leakage into the waste from the bottom (intake) end of the face. Air entry may also have occurred from old workings or airways at different levels or via a fault at the top end of the face.

Before the incident the results from the control points behind the face were satisfactory and gave no warning of any potential hazard. The very rapid deterioration of the situation highlights the potential seriousness of heatings.

The rapid application of large quantities of nitrogen and the modified mining program both contributed to the resolution of the incident. Historically, rates of increase of CO such as those observed would, without the injection of nitrogen, have forced the abandonment of the district. Injecting nitrogen via the bottom level allows all points in the waste to be accessed rapidly as was confirmed by the flow of SF_6.

In order to be able to tackle incidents of this type effectively it is essential to have a complete understanding of the flow pattern of nitrogen in the district and to be able to increase the flow rate rapidly using additional flexible lines if necessary.

A plan should be maintained on the surface showing the location of any existing water seals and sites for additional water seals. A plan of the mine should be available that shows all the old workings. Nitrogen suppliers must be capable of providing the contract quantities at short notice.

3. SECOND INCIDENT

The face continued working after the first incident without difficulty until early in June 1983 when traces of CO were found in the roof of the return entry of 66. Six holes were drilled and used to inject nitrogen in the roof of the roadway (Figure 6-2). The measurements indicated a steady level of 50 to 150 ppm of CO and 10% to 18% oxygen. On July 15 the Unor in the return of 66's showed an increase in

CO level. An inspection of 66's return at 4:30 pm revealed fumes in the roof. A Drager measurement indicated 3 000 ppm of CO. A little later flames were seen in the roof. The fire was surrounded by a glowing zone. At 7 pm nitrogen was put through the pipes in the roof at a rate of 150 m^3/h.. The glowing continued in the fire zone. By 930 pm the Unor in the return read 3 ppm having registered 13 ppm at about 5 pm.

On the third shift of July 15 the air flow on 66's face was reduced to 4.4 m^3/s. Additional sealing was carried out in the return using plastic netting as well as foam. The roof pipe was put under nitrogen. All the visible leakage points in the roadway were sealed. Continuous sampling was carried out on the roof pipes and in the return over three shifts. The flow of air to the fire zone was traced using SF_6. By July 19, at 5 pm the main return Unor registered 2.2 ppm. Holes 5/2 registered 6 700 ppm CO and 5% oxygen. Work continued on sealing the roadside byt by 10 pm the Unor reading had increased to 40 ppm. At 4:30 am 900 m^3/h of nitrogen were being introduced into the return roof pipes. At 5:30 am the Unor was reading 100 ppm. Smoke and fumes were present in the roof and floor.

At 5 am the decision was made to seal off the district by an hydraulic seal in the return. At 9 am a team of rescue men approached the fire through the face and saw burning material falling from the roof near roof hole 5. At 10 pm a second team was unable to pass the area near roof hole 1.

The decision was taken to inject 600 m^3/h of nitrogen in the bottom level of 66, 30 m from the face and to close the intake by a seal. The presence of more than 100 ppm of CO in the main return at the 320 m level forced the discontinuation of access to this level and to the #2 shaft from 9:30 am to 9 pm on July 20.

The CO readings in the face return reduced from 1400 ppm at 3 pm on the 20th to 85 ppm at 9 pm on the 20th and 2 ppm at 10 pm on the 21st. Values in the main return at 320 m were 140 ppm at 3 pm on the 20th, 70 ppm at 6 pm and 8 ppm at 9 pm. The hydraulic seal on the face return was completed at 4 pm on the 21st. The seal on the intake was completed at 4 pm on the 22nd. The water level of the seal in the return rose to the level of the fire at 4 am on the 23rd. Nitrogen flow into the bottom level of 66 was 350 m^3/h at 10 am on the 24th.

The intake seal was removed on August 17. Using auxiliary ventilation the top end of the face was reached on October 7. The new return connection was completed on October 12. Activities on face 66 restarted on October 14.

4. COMMENTS

The first incident illustrates the control of a heating in an inaccessible goaf area by injecting nitrogen. The regular supply of a controlled quantity of nitrogen and the systematic sampling in the goaf are of interest.

The second incident illustrates the difficulty of combatting localized fires, for example, in the roof of an entry with nitrogen, because of the difficulty in maintaining a sufficiently high concentration of nitrogen in the fire zone because of the dilution by air inevitable in such a situation. However, nitrogen injection into the area after sealing certainly played a part in expediting the recovery of the district.

FIGURE 6-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

LONGWALL 66 - ROZELAY MINE GOAF HEATING

FIRST INCIDENT

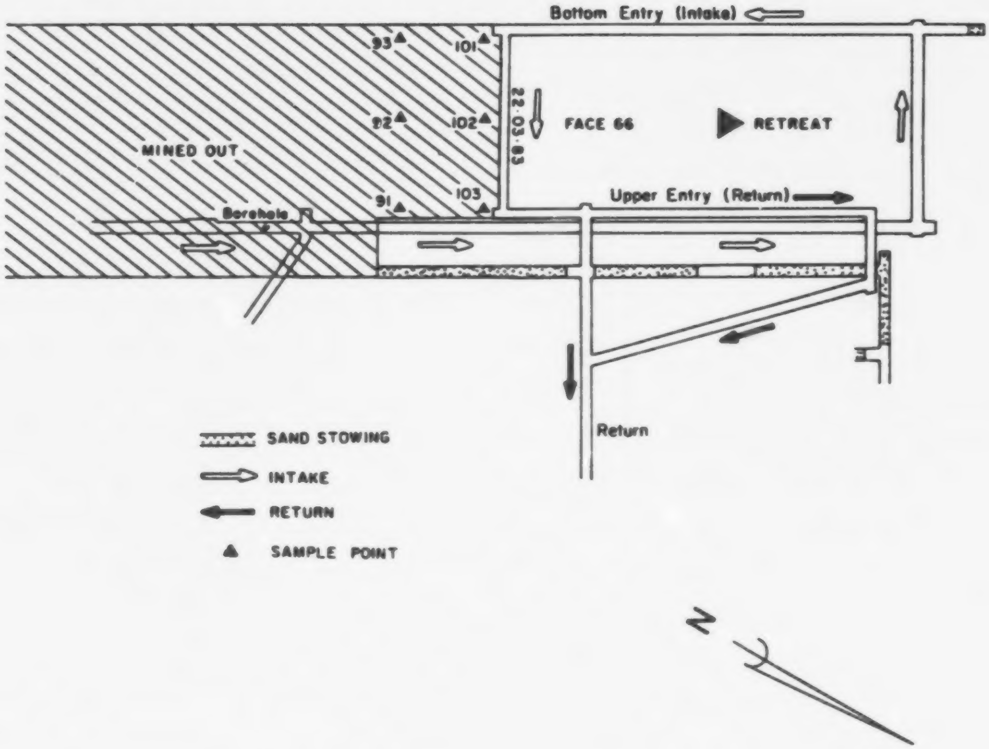


FIGURE 6-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

LONGWALL 66 - ROZELAY MINE FIRE IN RETURN
SECOND INCIDENT

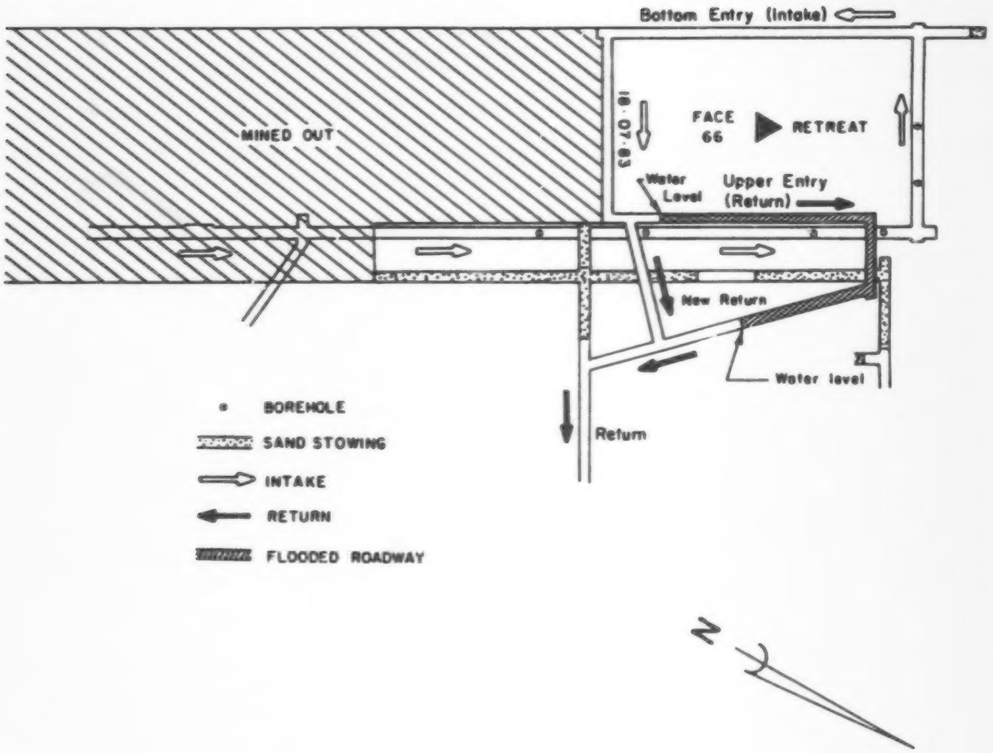
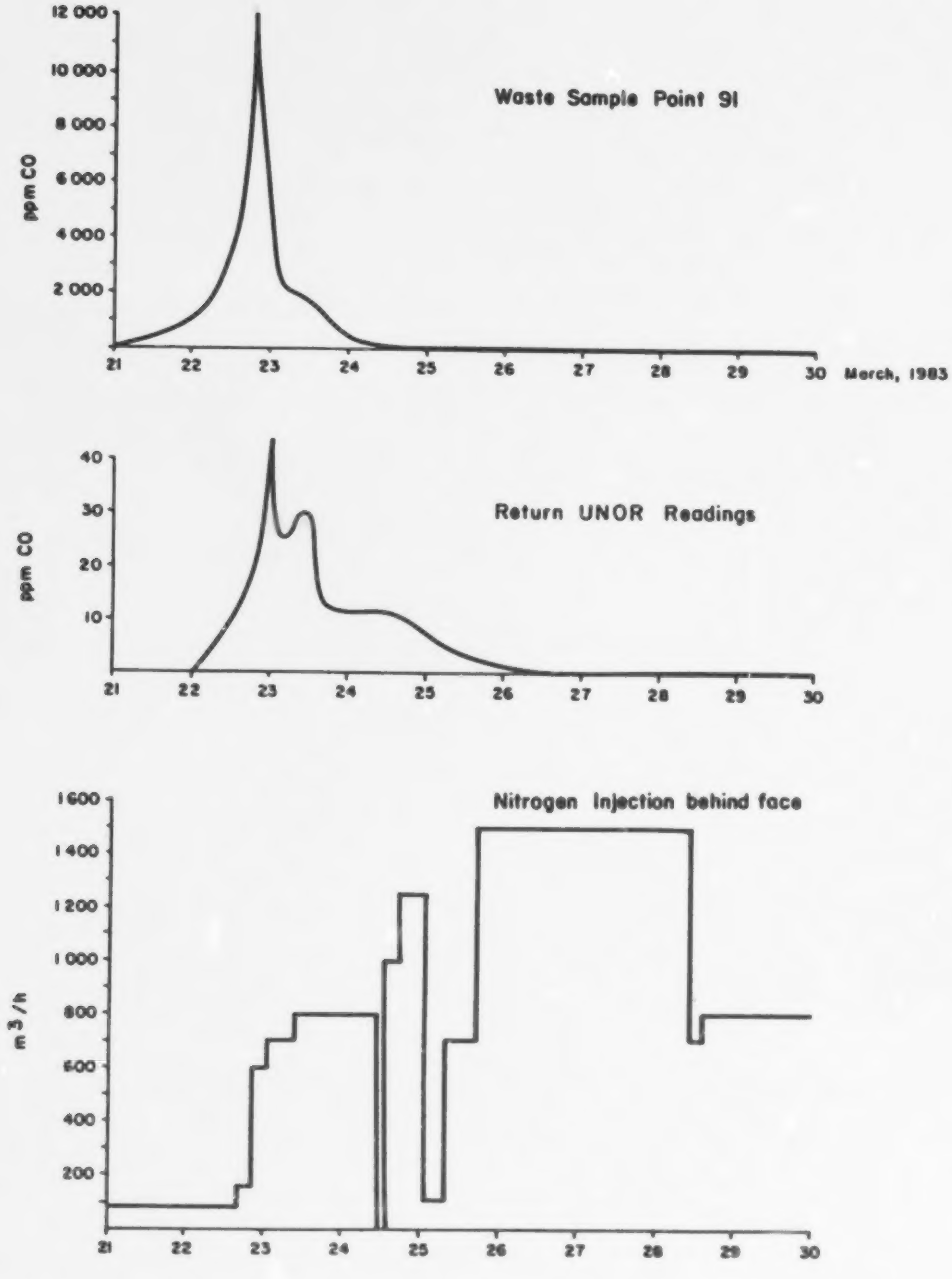


FIGURE 6-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

CARBON MONOXIDE AND NITROGEN MEASUREMENTS ROZELAY MINE



GERMANY

LUISENTHAL COLLIERY

TYPE OF MINING	Longwall
DATE OF INCIDENT	1975
TYPE OF FIRE	Inaccessible fire in gob or pillar
INERTING AGENT USED	Nitrogen gas
DELIVERY SYSTEM	4280 m existing methane drainage range
GAS DELIVERY RATE	10 - 70 m^3/min
TOTAL QUANTITY INERT GAS USED	556 024 m^3
SPECIAL FEATURES	Inerting used to control fire until face equipment could be salvaged and area sealed off
REFERENCE	Muller, Rudolf 1976: **Bekaempfung eines Verdeckten Grubenbrandes durch Inertisierung mit Stickstoff;** (Suppression of a Concealed Mine Fire by Inertization with Nitrogen), Glueckauf, v. 112/14, (Jul), p. 810-816.

1. BACKGROUND

The Alsbach District of the Luisenthal Colliery produces coal from 2 seams: Seam 2, 3.3 m thick, lies 20 m stratigraphically above Seam 3, 2.7 m thick. Coal is produced by longwall mining using the Z system. Panel 2A had been stopped short due to geologic problems; the face line in Panel 3A had been shortened for the same reason, but was ready to resume production. Panel 2B was producing coal advancing along strike.

CO concentration of 15 ppm were recorded in Panel 2B on September 23, 1975 (Figure 7-1). The source of the CO was found near Seal 395 in Haulage Road 72 (Figure 7-2). The fire, therefore, was thought to be in the gob or pillar of Panel 2A. No. 2 Seam tends to have a high spontaneous combustion potential. This hidden fire threatened both Panels 2B and 3A. These Panels represented a capital equipment value of DM 12 million.

At this time the main concern was to continue coal production from Panel 2B. The initial procedures included spraying and anhydrite coating on the walls of 2B Conveyor road for 20 m updip from Seal 395 and setting an air regulator in Haulage Road 71 to reduce the differential pressure across Seal 395. In case these measures would not control the fire, the following were initiated in order to isolate Panel 2B from the fire zone (Figure 7-3).

- drivage of the diagonal
- drilling 2 small diameter holes for fly-ash slurry pipelines
- drilling 2 large diameter holes to reduce the pressure across Seal 395 and to provide intake air if the diagonal was delayed
- site preparation for seals 450 and 451
- plan to fill headings between Seals 450 and 451 with fly-ash
- site preparation for seals 454 and 455 in case a larger area had to be contained

While this work was progressing the CO level (Figure 7-2) rose to such a level that Panel 2B had to be stopped on September 30. By October 13 the planned work had been completed and Panel 2B went back into production. However, after October 13, the fire shifted and began affecting Panel 3A. By October 15 it could be predicted that under existing conditions Panel 3A would have to be abandoned by October 20. The value of the face equipment was about DM 6 million.

At this point the prime objective became the salvage of the Panel 3A equipment. The decision was made to control the fire by inerting the gob of Panels 2A and 3A with N_2 until the salvage operation was completed and to seal off the area.

2. NITROGEN SUPPLY AND DISTRIBUTION

It was estimated that 65 000 m^3 nitrogen would be required initially followed by a continuous nitrogen supply of 5 to 30 m^3/min during a 3-week salvage operation for a total volume of between 220 000 m^3 and 970 000 m^3.

A steam locomotive was used as heat source for the gasifier. The distance for the nitrogen supply from surface to seal 395/450 was 4280 m. A methane drainage range was available for most of the distance, but had to be extended 720 m to Seal 450. The operating pressure for the drainage range was limited to 3.0 bar, allowing nitrogen to be delivered at a rate of 70 m^3/min.

3. OPERATING PROCEDURE

Inerting was started on October 18 with an N_2 flowrate of 70 m^3/min. Within 9 hours the CO concentration dropped to 25 ppm and the N_2 flowrate was dropped to 25 m^3/min (Figures 7-4 and 7-5). The N_2 flowrate was then continuously adjusted to maintain a CO level of

15 ppm. Salvage operations were started on October 19 and continued until November 5. During this time period the N_2 flowrate varied between 10 and 25 m^3/min. Seals 454 and 455 were closed on November 6.

4. COMMENTS

The costs for the inerting procedure are shown in Table 7-1. Some DM 400 000 were expended to recover DM 6 million of equipment - an excellent cost/benefit ratio.

Inerting at Luisenthal Colliery was successful in achieving the objectives. A concealed fire cannot be put out by this method, but it can be controlled so that other measures can be carried out.

The preparation time required to start inerting after the decision had been made was 3 days. Equipment should be made available to reduce this to 24 hours.

TABLE 7-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

INERTING COSTS

	Rent	Labour	Fuel and Materials	Transport	Total	%
Steam Locomotive	6 000	16 150	36 750	15 250	44 150	11.1
Gasifiers	1 650	14 700	700	5 800	22 850	5.8
Instrumentation		3 878	1 528		5 406	1.3
Nitrogen 556 024 m^3			259 300		259 300	65.4
Other Costs						
Surface		27 850			27 850	16.4
U/G		37 278			37 278	
Total	7 650	99 856	268 278	21 050	396 834	100

FIGURE 7-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

PARTIAL MINE PLAN, ALSBACH DISTRICT, LUISENTHAL COLLIERY

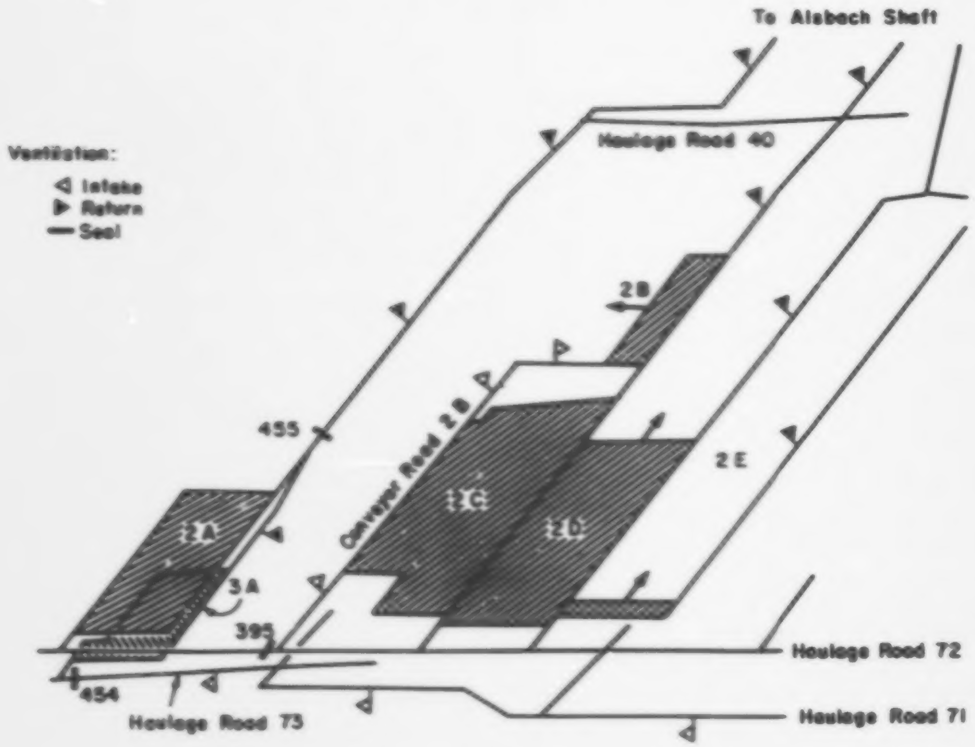


FIGURE 7-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

CO CONCENTRATION IN THE SECTIONS FROM PANEL 3A (1400 m^3/min) AND PANEL 2B (1250 m^3/min)

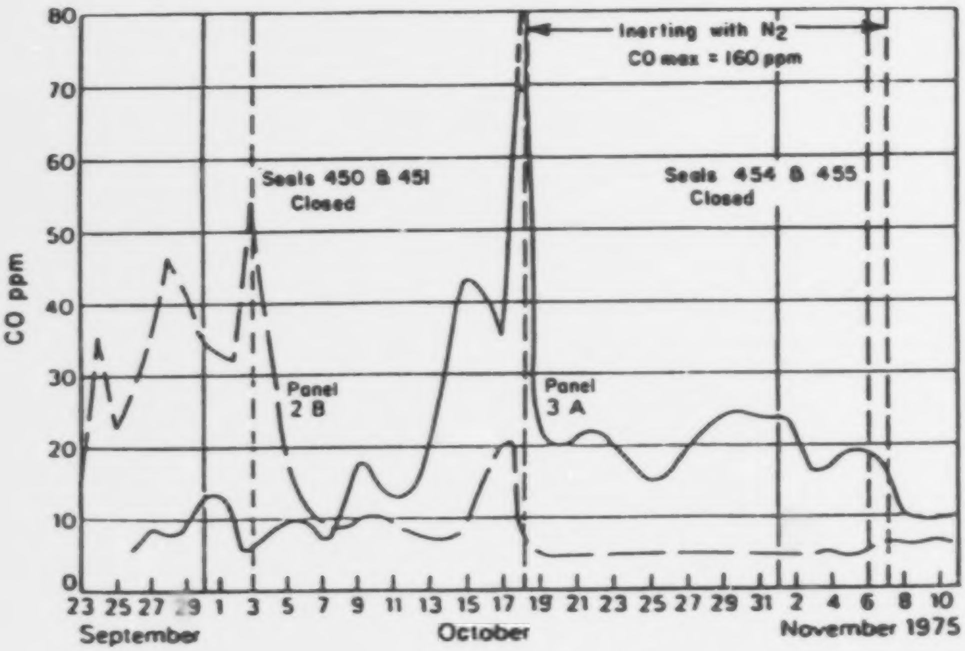


FIGURE 7-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

ADDITIONAL CONTROL MEASURES NEAR BOTTOM OF CONVEYOR ROAD 2B

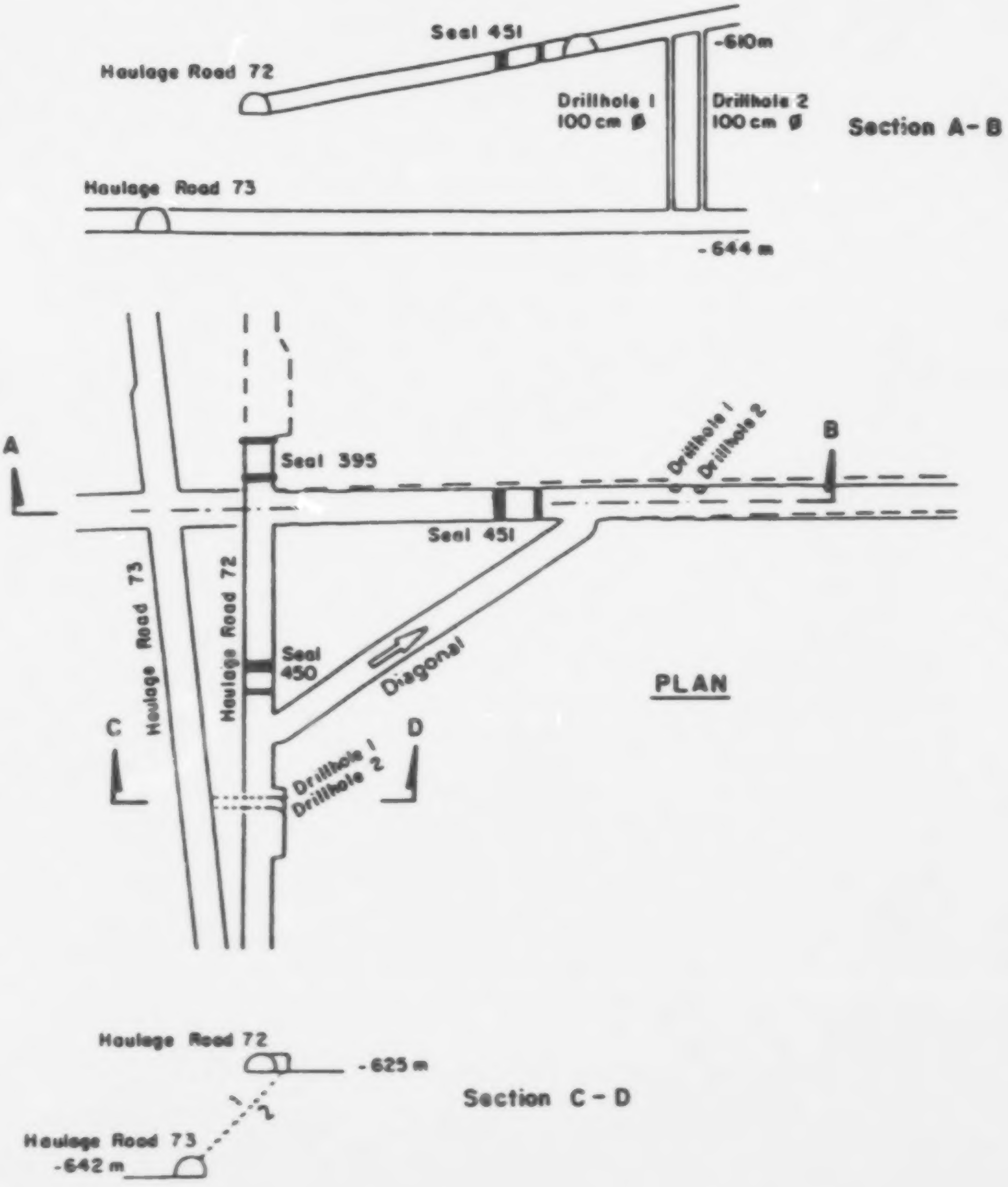


FIGURE 7-4
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

NITROGEN SUPPLY AND CO/CH_4 CONCENTRATION IN THE PANEL 3A RETURN

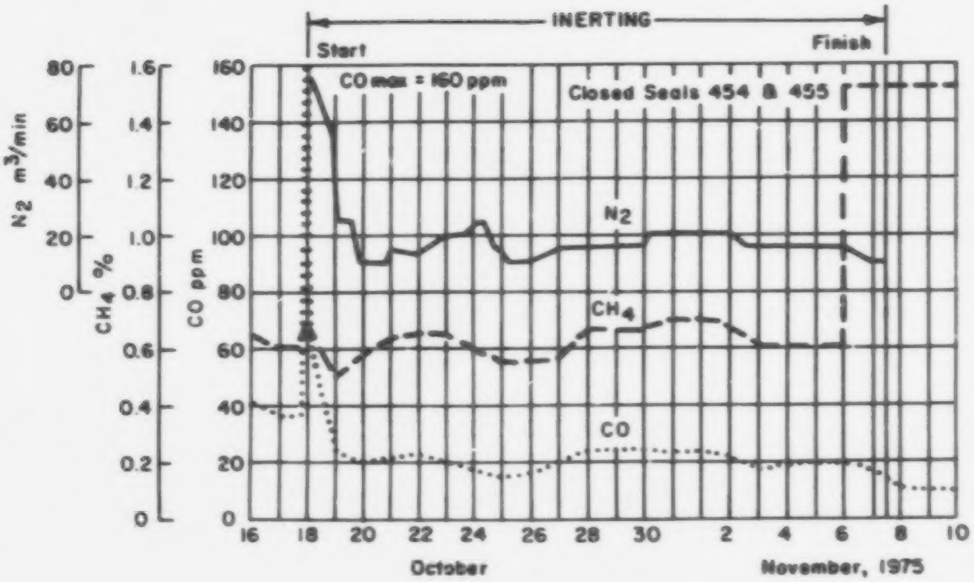
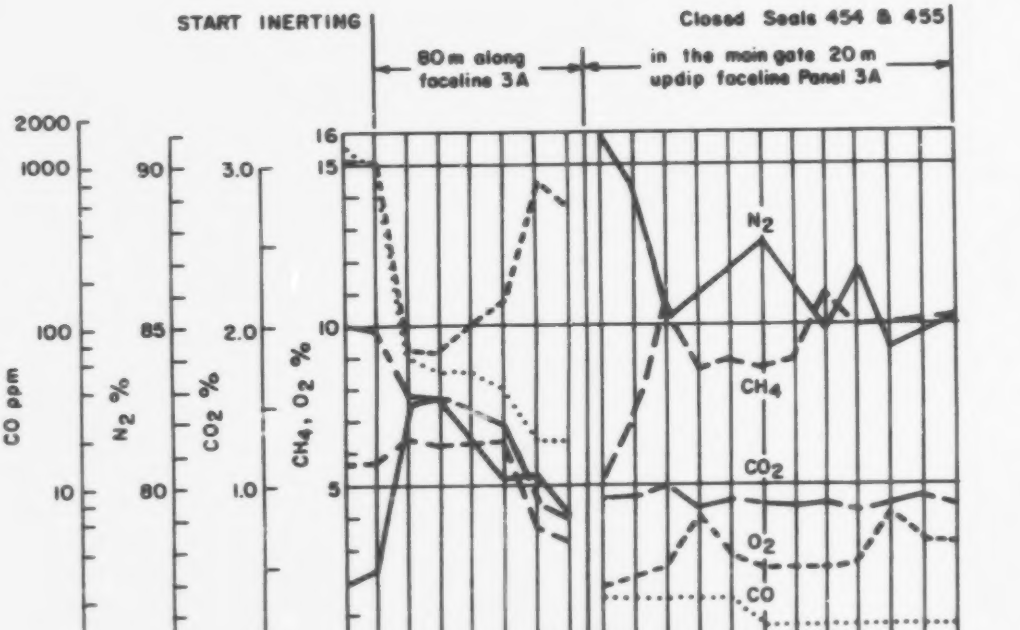


FIGURE 7-5

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

RETURN AIR COMPOSITION IN PANEL 3A DURING INERTING



GERMANY

OSTERFELD COLLIERY

TYPE OF MINING	Longwall
DATE OF INCIDENT	1974
TYPE OF FIRE	Heating in goaf of advancing longwall
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Liquid nitrogen vaporized on surface and delivered by pipeline to face
GAS DELIVERY RATE	50 - 100 m^3/min
TOTAL QUANTITY INERT GAS USED	154 000 m^3
SPECIAL FEATURES	First use of nitrogen for controlling mine fires in Germany
REFERENCE	Kugler, U. and Schewe 1975: **Control of a Mine Fire at the Osterfeld Colliery by the Introduction of Nitroghen;** Special (English) offprint, 23 p., from Glueckauf, v. 3/10, p. 467-472.

1. BACKGROUND

Panel 1 East in the 2.6 m thick Gustav seam in Osterfeld Colliery was worked by a 190 m long advancing longwall face, equipped with double-ended shearer-loader and shield supports. Anhydrite roadside packing was used at both face ends. An overthrust fault in the line of the face limited panel advance. (Figure 8-1).

Shortly before reaching the planned limit of the working CO was detected in the drained gas, which was traced to spontaneous combustion in the caved area. Coal getting ceased on November 6, 1974. Removal of the shield supports from the face commenced on November 11, 1974 with the CO level in the return air at 28 ppm with an air flow of 22.5 m^3/s, and was completed on November 27, 1974. Stoppings 1 and 2 (Figure 8-1) were installed under normal ventilation conditions and the access tubes closed simultaneously on December 1, 1974. The siting of Stopping 2 was selected to allow the gateroad to be re-used for the adjacent Panel 2 East. Auxiliary forced ventilation was used in this gateroad.

It was anticipated that the high gas emmission in the sealed off area would extinguish the spontaneous combustion. As a precaution, the new panel would not commence production until the CH_4 level in the sealed off panel had exceeded the upper explosive limit.

Before this condition was reached, two small explosions occurred on December 2, 1974, which resulted in a temporarily increase in CO and the auxiliary ventilation system of the old conveyor road being put out of action. A more serious explosion on December 3, 1974 needed further action. Stoppings 3, 4 and 5 were prepared but not closed. The air from the old conveyor road had been freshened up by the ventilation of the new face, 2nd panel east, and showed a CO level of 60 ppm on reaching the staple pit. Two small deflagrations occurred but a regular pattern could be detected in the flow. It was decided to install stopping 2a in the old conveyor road as soon as conditions free of explosion hazard could be created. The only reliable means of achieving this was by introduction of large quantities of inert gas. It was decided to introduce nitrogen into the combustion zone from a point near stopping 2 in the direction of the pressure gradient. Immediately following a deflagration the risk of sending a rescue team into the old conveyor road for a limited period to connect the nitrogen supply lines would be minimized.

The erection of Stopping 2a could then be carried out under protection of the inert gas. Analyses were to be made to determine whether the total length of the old conveyor road was free from explosible gas mixtures.

2 NITROGEN SUPPLY AND DISTRIBUTION

Arrangements were made with Messer-Griesheim GMBH to supply up to 600 000 m^3 of nitrogen from their Duisburg plant. The gas was delivered by 13 t tankers in liquid form and passed through evaporators to give a controlled rate of between 50 to 100 m^3/min at 2 - 7 bar and 0°C.

The evaporation plant consisted of a 6.5 m^3 intermediate tank into which the liquid gas, cooled to -196°C was fed from the tank truck, through a reducing valve. The liquid nitrogen passed from this tank to a cold evaporator then to 3 gas evaporators each producing 33.3 m^3/min, which were heated using colliery steam. The required gas pressure was obtained from the evaporation process without using a compressor and the outlet temperature of the gas was between 10°C and 20°C.

Gas conditions were monitored as follows:

- flowrate using a single tap orifice gauge
- pressure with a Bourdon-tube gauge
- temperature with a platinum resistance thermometer.

The plant was manned by 2 persons on a 24-hour basis. The gas was piped to the face through an existing 100 mm diameter pipeline to the staple pit, connected up to a second existing 100 mm diameter pipeline in the old conveyor road, previously used for anhydrite pump packing. Total pipeline distance was 3761 m.

3. OPERATIONAL PROCEDURE

On December 6, 1974 the pipework was connected up, cleared of air and nitrogen fed into the face end at the rate of 30 m^3/min increasing to 60 m^3/min. A remote sampling line was laid.

Figure 8-2 shows the composition of the gas samples taken from east of Stopping 2a and their relationship to the explosive triangle. The effect of a nitrogen supply interruption for 3 hours on December 7, 1974 is clearly shown. Construction of stopping 2a commenced on December 7 after more than 40 000 m^3 of nitrogen had been fed in, with the installation of an inflated air-bag seal. The air behind this temporary seal became totally inert allowing construction of the permanent stopping by December 9, 1974. The nitrogen supply was reduced, then cut off on December 11, 1974.

The adjacent workings recommenced production on December 9, 1974.

A total of 154 000 m^3 of nitrogen were used.

Figure 8-3 shows the development of the fire in terms of CO generation and the quantity of nitrogen added for control. The negligible CO level after December 11, 1974 indicated that the fire had been extinguished throughout the entire combustion area.

4. COMMENTS

This example illustrates the inerting of a goaf area in which there was a well established heating. Because of the extent of the heating, large quantities of nitrogen were employed, which effectively controlled the fire. The use of nitrogen greatly assisted the activities of the rescue teams in entering the area to erect seals.

FIGURE 8-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

PLAN OF WORKINGS IN THE GUSTAV SEAM OSTERFELD COLLIERY

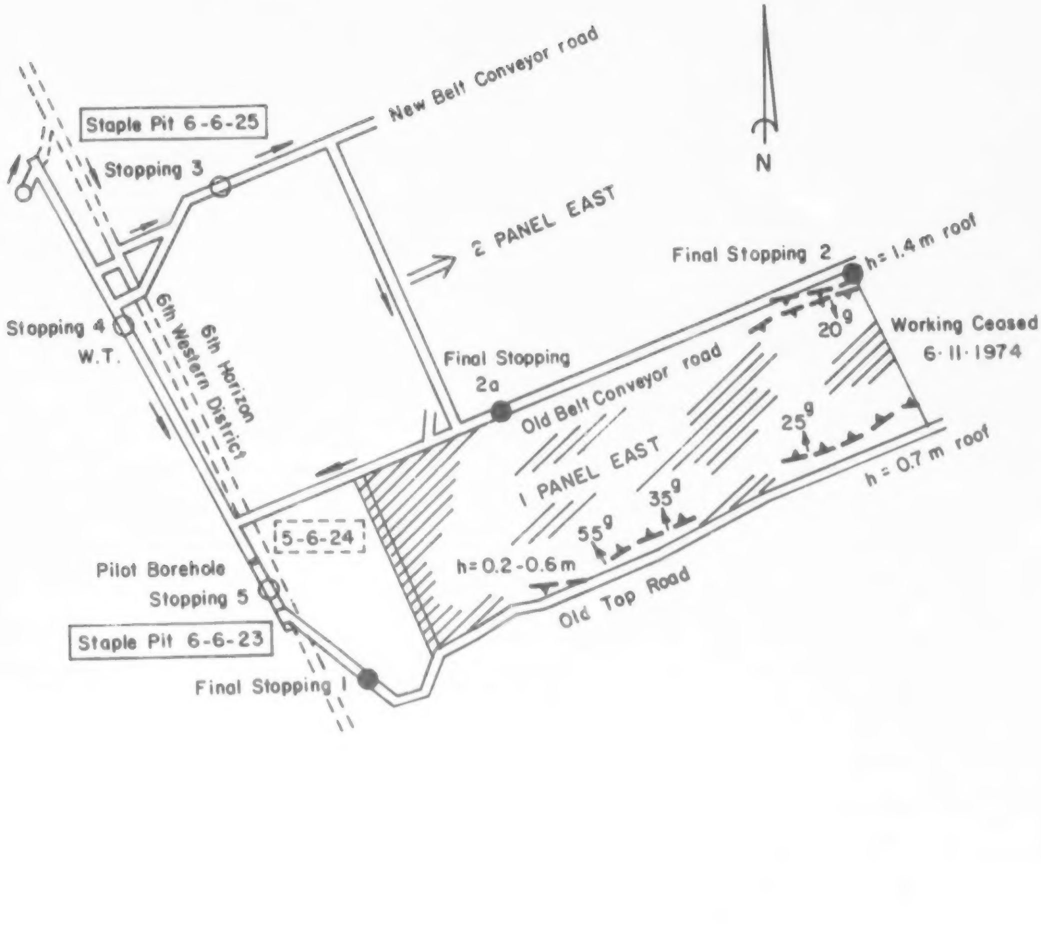


FIGURE 8-2

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

THE COMPOSITION OF THE GAS SAMPLES TAKEN FROM EAST OF STOPPING 2a AND THEIR RELATIONSHIP TO THE EXPLOSION TRIANGLE

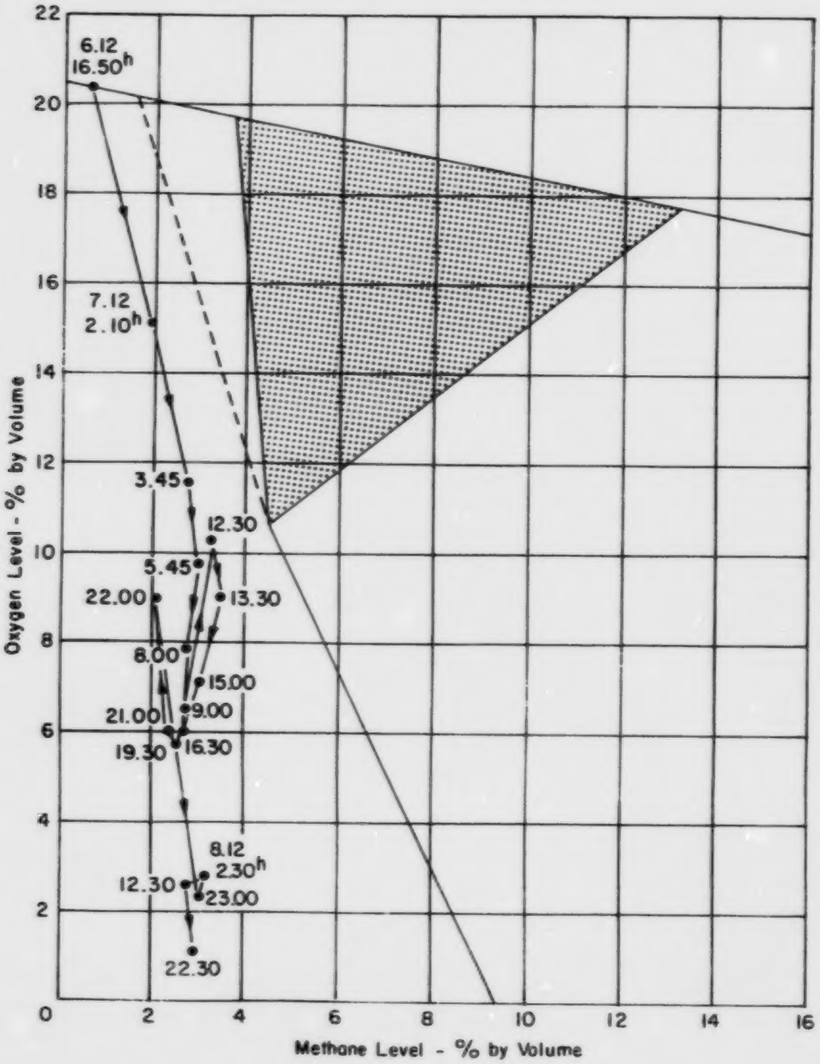
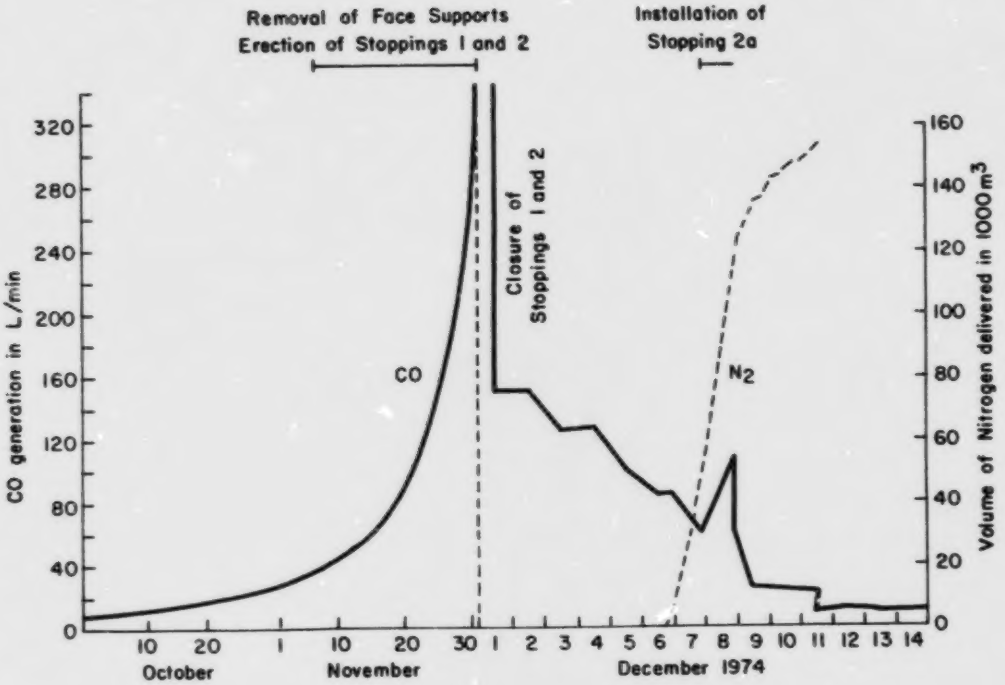


FIGURE 8-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

CO GENERATION AND QUANTITY OF NITROGEN FED IN DURING OPERATION TO CONTROL THE FIRE IN THE OSTERFELD COLLIERY



GERMANY

SCHLAGEL UND EISEN COLLIERY

TYPE OF MINING	Longwall
DATE OF INCIDENT	July 1976
TYPE OF FIRE	Methane gas ignited by electrical short-circuit
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Existing pipelines and hoses from surface. Single probes into fire site
GAS DELIVERY RATE	50 m^3/min reducing to 20 m^3/min
TOTAL QUANTITY INERT GAS USED	126 000 m^3
SPECIAL FEATURES	The source of repeated gas ignitions was extinguished by local nitrogen flooding; no seals were used
REFERENCE	Genthe, M. and Schonfeld, H. 1977: **Die Bekampfung Eines Offenen Grubenbrandes mit Objektinertisierung Durch Stickstoff;** (Fighting an Open Mine Fire by Target Inerting with Nitrogen), Glueckauf, v. 113/8, (Apr), p. 407-411.

1. THE INCIDENT

A methane gas ignition occurred at 04:30 h on July 8, 1976 in Panel 4 of the Gustav Seam in the Schlagel and Eisen Colliery. The initial ignition was caused accidently by a shorted electrical circuit near the stage loader in the conveyor gateroad and extended for about 10 m along the roof of the gateroad. This ignition caused no appreciable damage. However, a fire was ignited between the anhydrite pack and the gateroad supports which caused repeated secondary ignitions at intervals of 3 to 5 seconds. All secondary ignitions took place between roadside supports and the anhydrite pack within 8 m of the initial ignition. Flame propagation was relatively slow (non-explosive) towards the face line. The site of the actual fire could not be determined accurately. No injuries occurred.

2. BACKGROUND

The site of the fire was located in the Gustav Seam workings, South District of the Colliery. No. 4 Panel (Figure 9-1) is a longwall face advancing downdip; the face pitches 8° to the southwest. The Gustav Seam is 1.9 m thick and the face length is 210 m. Both gateroads are supported by yielding steel arches on 0.75 m centres covered by wire mesh and brattice cloth. The intake gateroad has a foam-filled wood pack, whereas the conveyor gateroad is sealed off by a 1.3 m anhydrite pack.

The gateroads are maintained at a nominal cross-sectional area of 16 m^2. The fresh air supply is 1600 m^3/min at a pressure differential of 39 mm WG between face ends. Methane drainage holes are drilled into the roof and floor from the conveyor gateroad. These are connected to a collection range at 10 kPa below atmospheric pressure and produce, on average, 45 m^3/min. In spite of this methane drainage, the return air averages 1% CH_4 by volume with peaks up to 1.5%. Because of this high methane concentration, Panel No. 4 operates under an exemption from the Mine Regulations.

Panel production is 1600 t/day with an average face advance of 3.2 m/day. Ventilation flow is in the same direction as the coal transport; up the coal face at 8° and out along the conveyor gateroad.

The gate conveyor had been extended on the night of the accident; while the stage loader was being moved up a cover plate caught the cable between the transformer and the switchgear and pulled it out of the transformer. The resulting spark caused the initial methane ignition.

3. FIRE FIGHTING PROCEDURES

The whole South District, comprising three longwall panels, was closed down; only mine rescue teams were allowed to carry out work in this area. The fire causing the secondary methane ignitions was thought to be:

- either a gas bleeder burning like a bunsen burner
- or burning combustible material, e.g. wood behind the supports.

Both possibilities did not lend themselves to conventional fire fighting procedures. Nevertheless, water from two ranges was injected into the hand stowed area behind the gateroad supports through 18 probes driven into the gob-side rib. As had been anticipated, there was no effect on the frequency of methane ignition. However, it was felt that the cooling effect around the fire site would be beneficial.

In order to reduce the methane flow to the fire site a methane drainage hole was drilled from the relative safety of the face line. It was angled up into the roof strata over the fire. This borehole was connected to the methane drainage range in the early morning of July 9, producing 0.6 m^3/min of CH_4. This additional methane drainage was sufficient to increase the interval between ignitions to 5 - 6 hours.

The incombustible materials used in the support of the conveyor gateroad and the roadside pack showed their worth in this situation. After more than 24 hours of ignitions there was no detrimental effect on the supports. Had wooden lagging been used, the gateroad would probably have collapsed after this time interval.

Independant of the fire fighting activities, sealing the panel had been considered in the very early stages of the incident. Sites for the seals were selected and initial site preparation was carried out. However, sealing the panel was considered as a last resort.

4. TARGET INERTING WITH NITROGEN

Up to the time of this incident, six mine fires in Germany had been successfully put out by inerting into a sealed area. Because of the lengthy waiting period between inerting and reopening the district once sealed, the management decided to try 'target inerting' for the first time. This method requires that the location of the fire be known, since the fire source needs to be flooded with nitrogen to extinguish the fire. Where extensive glowing embers are present this method will not put out the fire source.

Inerting had been considered since noon July 8. A propane fired nitrogen gasifier had been installed (capacity 100 m^3/min) along with two air gasifiers, and liquid nitrogen storage was put in place. The delivery system for the nitrogen comprised (Figure 9-2):

- water range down the shaft
- hose connection

- water range along ventilation incline
- hose connection through No.3 Split
- anhydrite pipeline along conveyor gateroad
- six nitrogen injection probes.

800 m of newly developed inert gas hoses were used to connect the existing pipe ranges. The injection probes had been made for this application; they were driven into the gob-side rib. At 08:00 h on July 9, nitrogen was delivered to the fire site at 50 m^3/min. This procedure gave the desired results. Only one small ignition took place after inerting was started. The nitrogen delivery was soon reduced to 40 m^3/min.

The roof and rib support between arches was removed under the protection of continued nitrogen flow. No glowing embers of any kind were found. The ignition source must have been a gas bleeder.

Repair work continued in the gateroad while injection of nitrogen continued until 08:00 h July 12, although the nitrogen delivery rate had been dropped to 20 m^3/min on July 10. During inerting there was never an oxygen deficiency in the return air that could have been detrimental to human beings.

The face was restarted on the23:00 h shift on July 12, 1976. A total of 126 000 m^3 of Nitrogen were used, at a cost excluding mine labour, of CDN $30,000.

FIGURE 9-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

GUSTAV SEAM NO.4 PANEL PLAN

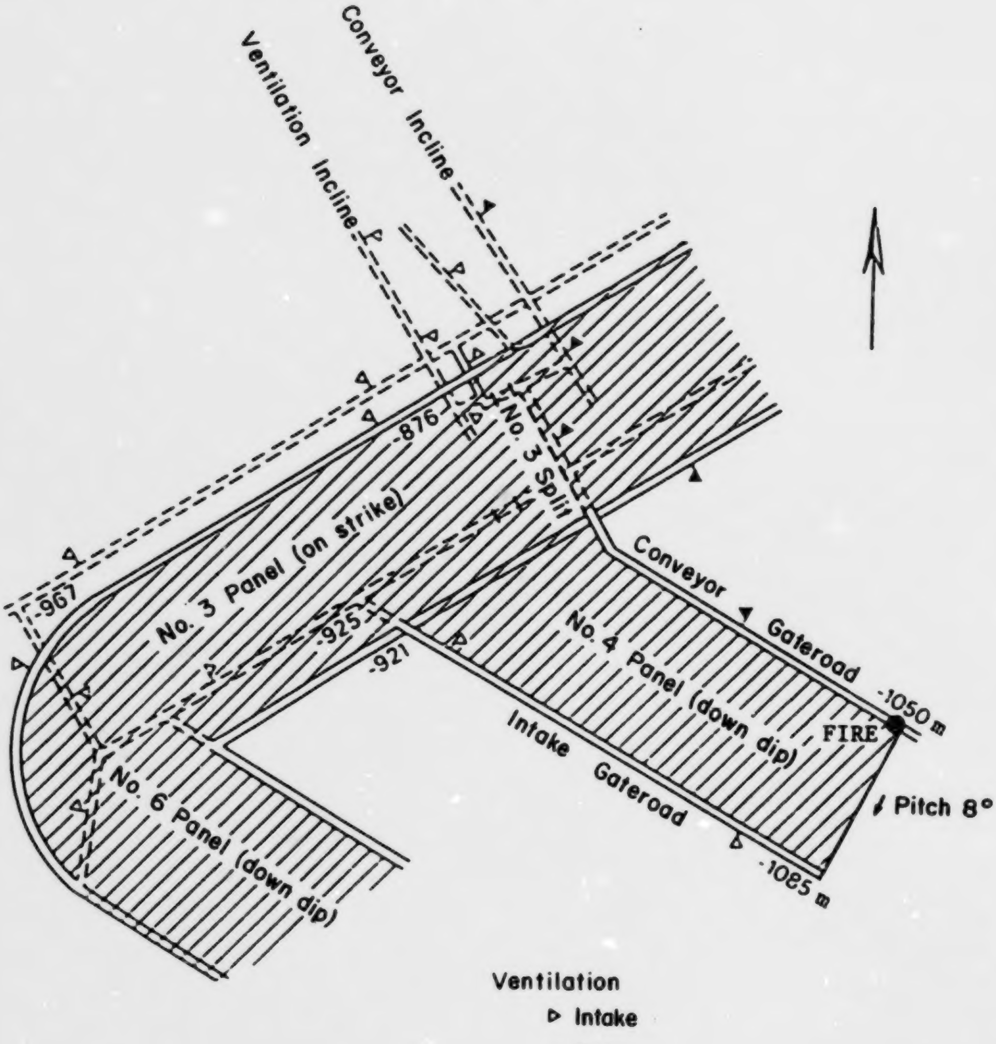
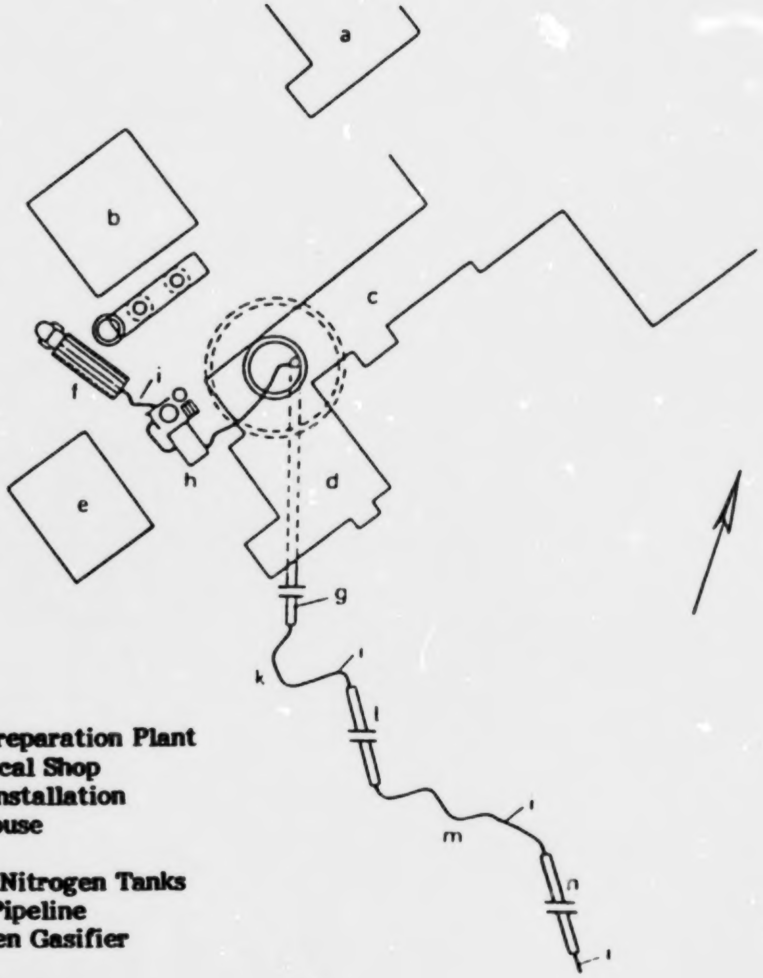


FIGURE 9-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

NITROGEN SUPPLY PIPING PLAN

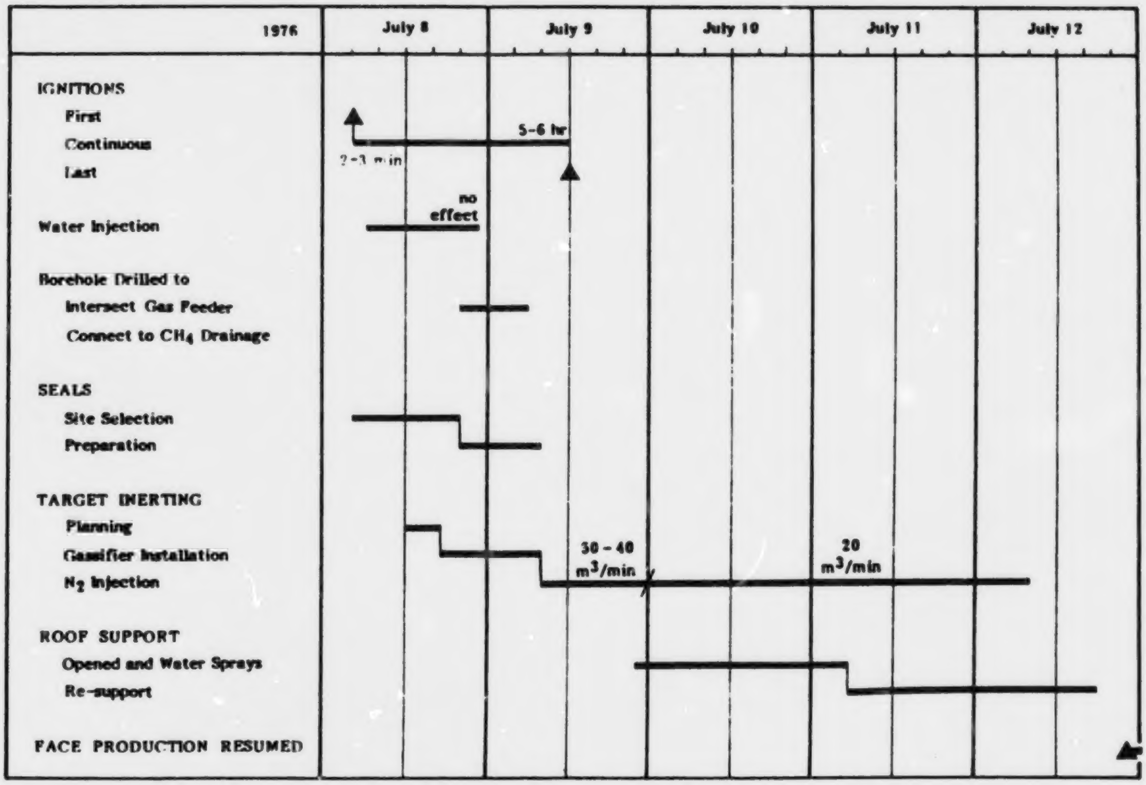


a. **Coal Preparation Plant**
b. **Electrical Shop**
c. **Shaft Installation**
d. **Warehouse**
e. **Hoist**
f. **Liquid Nitrogen Tanks**
g. **Shaft Pipeline**
h. **Nitrogen Gasifier**
i. **Hose**
k. **Shaft Bypass**
l. **Ventilation Incline**
m. **No. 3 Split**
n. **Gate Conveyor Road**

FIGURE 9-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

TIME SCHEDULE



INDIA

LAIKDIH DEEP COLLIERY

TYPE OF MINING	Multi-slice room-and-pillar with sand stowing
DATE OF INCIDENT	1981
TYPE OF FIRE	Active fire in worked out area initiated by spontaneous combustion
INERTING AGENT USED	Products of combustion
DELIVERY SYSTEM	Inert gas generator on surface. Gas delivered by pipeline to fire area
GAS DELIVERY RATE	8.5 m^3/min
TOTAL QUANTITY INERT GAS USED	100 000 m^3 inert gas up to August 1981 - ongoing
SPECIAL FEATURES	Use of inert gas generator located on surface to control long-standing fire in sealed area.
REFERENCE	Anon 1981: **Interim Report of Inert Gas Infusion at Laikdih Deep Colliery;** Cent. Mine Plan. & Des. Inst. Ltd., Min. Tech. Div., Inter. Rept. (Dec), 23 p. + 2 Annexure and 4 Plates.

1.0 BACKGROUND

The Laikdih Deep Colliery is located near Barakar in India. The 27 m thick Laikdih Deep seam is mined in three sections with stowing, leaving partings of Thama Coal between the sections. Each section is mined in two or more lifts of 2.4 m each.

During depillaring of the main rise bottom section in 1958, a fire was observed and the area sealed off.

Despite the various efforts to contain it, the fire continued slowly advancing towards the main intake roadway, threatening the life of the mine.

The Central Mine Planning and Design Institute Ltd. (CEMPDI) were consulted in 1981 and they decided to attempt to control the fire using an inert gas generator they had procured.

2.0 SUPPLY AND DISTRIBUTION OF INERT GAS

Inert gas was produced by an inert gas generator (manufacturer unknown), located on the surface of the mine. The generator (Figure 10-1) produced gas at the rate of 500 m^3/h at 5 m water gauge pressure and 35°C maximum temperature. It was passed through 2000 m of pipeline (150 mm diameter) through a stopping into the sealed fire zone.

3.0 OPERATIONAL PROCEDURE

Figure 10-2 shows the area of the mine in which the fire is active. It is contained to the north of the 2 East level by a row of stoppings in 3 section. Several stoppings are flush with 2 East level leaving no space for further strengthening. The fire had threatened the 2 East level on several occasions in 1980, and since this was the main intake railroad, the survival of the mine was at stake. The area around #27 and #30 stoppings was most seriously affected.

The following fire fighting measures were taken:

- strengthening of stoppings
- brick walling of pillar sides
- cement injection of the parting and pillar sides
- injection of CO_2 (270 cylinders)
- water infusion into the pillars and behind the stoppings
- superimposition of stoppings.

These measures had limited effect on controlling the fire.

The inert gas generator was commissioned at the end of March 1981. The inert gas was fed via an existing 150 mm diameter pipeline from the surface, down No. 2 Shaft and 2 East level to #27 stopping. The flow of gas was controlled by a valve at the doutlet pipe of the inert gas generator and by a second valve at the inlet pipe to the stopping. Sampling points were included.

The air/fuel ratio to the inert gas generator was controlled so that the resultant inert gas contained no CO and less than 1% O_2. A typical analysis of the inert gas would be:

15% CO_2, 0.6% O_2. Trace CO, 84.4% N_2

Inerting through stopping #27 commenced on April 2, 1981. The generator was operated for approximately four hours per day until May 26, 1981, and again from July 21, 1981 to August 10, 1981. In between the generator was stopped due to minor defects, non-availability of water or diesel fuel. From August 11, 1981 to September 27, 1981 the generator was under major repair, but has been operable since that time.

At the same time the inert gas was infused, water was continually injected through stopping #30.

The air analysis taken from the stoppings indicated that it was not possible to create an inert atmosphere behind all the stoppings. Some of the stoppings continued to contain above 19% O_2 and so were potential sites for future heatings.

The reason it was not possible to inert the sealed off area included:

- the inert gas was fed through only #27 stopping, and may have been unable to reach several areas due to roof falls and stowing
- the generator was operated for a maximum 6 hours per day. The 3000 m^3 of gas injected represented only 1% of the void in the sealed area
- despite sealant treatment and cement injection, there was a replacement of inert gas by air through the stoppings.

The use of inert gas effectively controlled the fire, but it was recognized that it would restart unless inerting continued or further measures were taken. The following steps were recomdmended:

- stop all leakages through stoppings or surrounding strata
- feed inert gas through several stoppings rather than a single one
- attempt to feed inert gas into the sealed area continuously rather than 4 hours per day. Case must be taken to ensure the general mine atmosphere remained safe
- once complete inertization of the area behind the stoppings is achieved, only intermittent feeding of inert gas will be necessary.

The cost of using nitrogen was calculated to be from 3 to 6 times greater than use of the inert gas generator.

4.0 COMMENTS

This Case Study demonstrates how the infusion of inert gas at Laikdih Deep Colliery helped contain a fire which had been gradually advancing since 1958 and by 1981 was threatening the main arterial road. It was considered that the fire could be extinguished if the inert gas could be fed continuously into the sealed area for 4 - 6 weeks.

The use of an inert gas generator located on the surface of the mine is demonstrated.

Where injection of inert gas is required over a relatively long period, this method has been shown to be more economic than the use of nitrogen.

FIGURE 10-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

SCHEMATIC FLOW DIAGRAM FOR INERT GAS GENERATOR

STREAM No	①	②	③	④	⑤
FLUID	KEROSENE H.S DIESEL	COMBUSTION AIR	COOLING WATER	DRAINED WATER	INERT GAS
FLOW RATE	50 KGS/HR	500 NM³/HR	20 M³/HR	20 M³/HR	[illegible]00 NM³/HR
PRESSURE	7.0 KG/cm² G	6,500 mm WG	3 KG/cm² G	UNDER GRAVITY	5000 mm WG
TEMPERATURE	AMBIENT	80° C (MAX)	35° C (MAX)	50° C	40° C

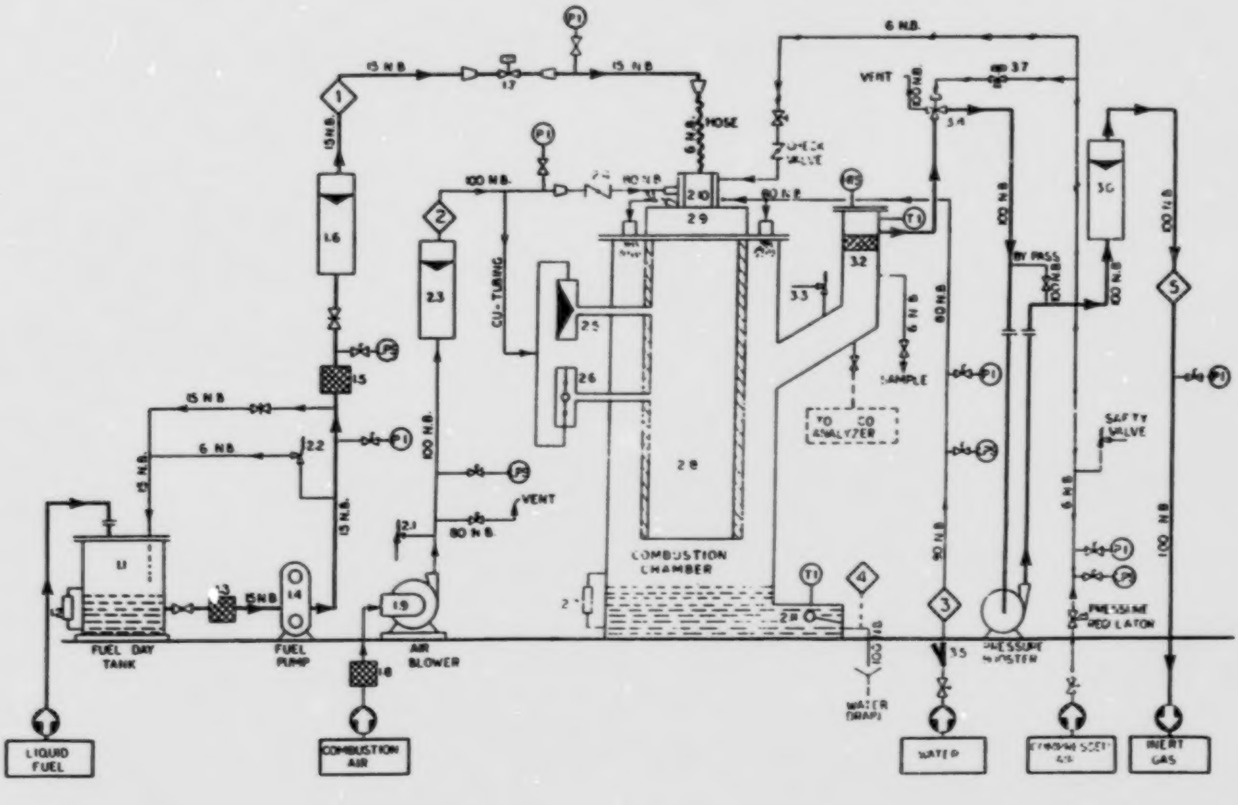
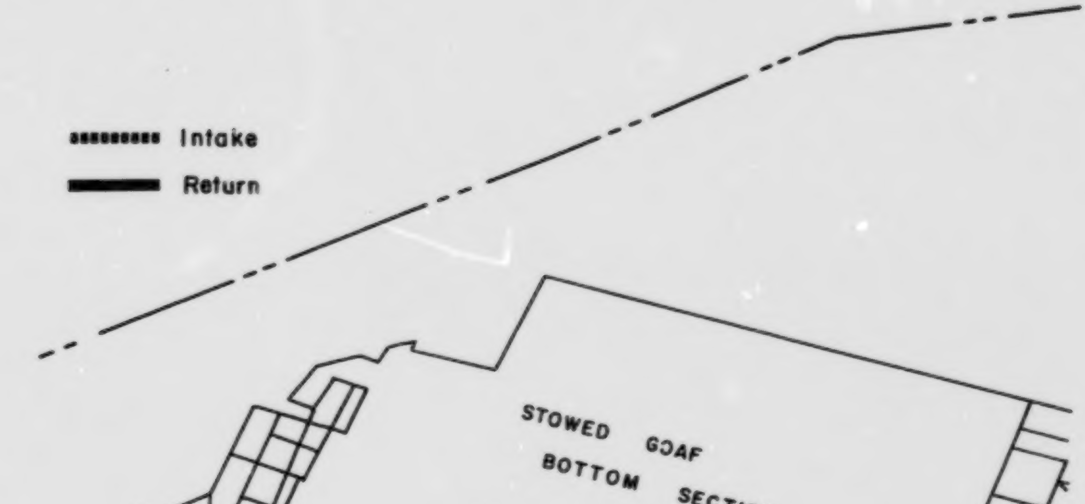


FIGURE 10-2

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

MINE LAYOUT - LAIKDIH DEEP COLLIERY



POLAND

BRZESZCZE MINE

TYPE OF MINING	Longwall
DATE OF INCIDENT	1978
TYPE OF FIRE	Ignition during blasting operation
INERTING AGENT USED	Inert gas
DELIVERY SYSTEM	CMI inert gas generator
GAS DELIVERY RATE	1200 m^3/min
TOTAL QUANTITY INERT GAS USED	330 000 m^3
SPECIAL FEATURES	Use of inert gas generator and cooling of fire zone by recirculation
REFERENCE	Kukuzka, A. - Personal communication: **Application of Gas Inerting for Control of Fire in Brzeszcze Coal Mine.** Poland.

1. BACKGROUND

On April 21, 1978 a mine fire developed during blasting operations in the Brzeszcze coal mine in the gassy 352 seam. Due to combustion of methane in the longwall face attempts to extinguish the fire by use of active means failed and the area had to be sealed off.

The fire area whose volume was estimated to be 150 Mm^3 was ventilated during the construction of gypsum fire dams 1, 2, 3 and 4 (Figure 11-1).

2. INERT GAS SUPPLY AND DISTRIBUTION

A CMI (GAG) inert gas generating unit was installed ahead of fire dam 1.

3. OPERATIONAL PROCEDURE

The fire dams were constructed and the CMI inert gas generator installed. The CMI unit was run for 4-1/2 hours, inerting the gases in the fire area and allowing the safe closing of dams 2 and 3. The total volume of inert gas fed into the area was 330 Mm^3. With the fire area sealed and inerted the fire was extinguished. However, the temperature inside the fire area was too high to open the fire dams, so a system for recirculation and cooling of the fire gases was installed (Figure 11-2).

A plugged duct section (800 mm dia) provided in dam 1 and a plugged duct section (300 mm dia) in dam 3 were prepared for connection with the duct line for recirculation of the combustible gases.

The duct line contained for following elements:

- two water coolers equipped with water jets and a drain water trap
- a control gate for control of gas flow rate to the cooling capacity of the system
- a booster fan for recirculation of inerted fire gases
- a water cooler
- gas sampling point
- temperature and flowrate measurement

The gases in the fire area, both beyond dam 1 and outlet dams 1, 2, 3 and 4 had been inerted by application of the CMI unit and contained 5% O_2 and above 80% CH_4.

In order to start the recirculation of inert gases and the cooling of the fire area, the following procedure was followed:

- the gates in the ductline'L' were opened and gases from the fire area were introduced into the ductline simultaneously with starting of the cooling system
- the control gate opened the ductline up to half of its cross section
- booster fan was started and the flowrate of gases adjusted.

The temperature of the recirculated gases dropped from 120°C to 35°C.

It was then necessary to reventilate the fire zone to remove the high concentration of methane. The methane drainage system for the area was started and a recirculation system installed for ventilating the area without opening the dams (Figure 11-3). A gas hazard zone was established. All men were withdrawn and electric power disconnected.

The reventilation procedure was as follows:

- ductline in dam 1 and 2 were closed
- a duct section was removed directly beyond the auxiliary fan
- air flowrate of 1200 m^3/min was maintained in incline drift 3

- the auxiliary fan 'W' was and gates opened

The fire area was then ventilated at a rate of 400 m^3/min for a period of 25 minutes.

After a further 4 hours, dams 1, 2 and 3 were opened and mining operations restarted in the area.

4. COMMENTS

This case study demonstrates how the use of the CMI inert gas generator allowed safe sealing of a methane fire. The methods used for cooling and reventilating the fire zone allowed rapid resumption of production.

FIGURE 11-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

VENTILATION SCHEME DURING FIRE FIGHTING OPERATION
BRZESZCZE MINE

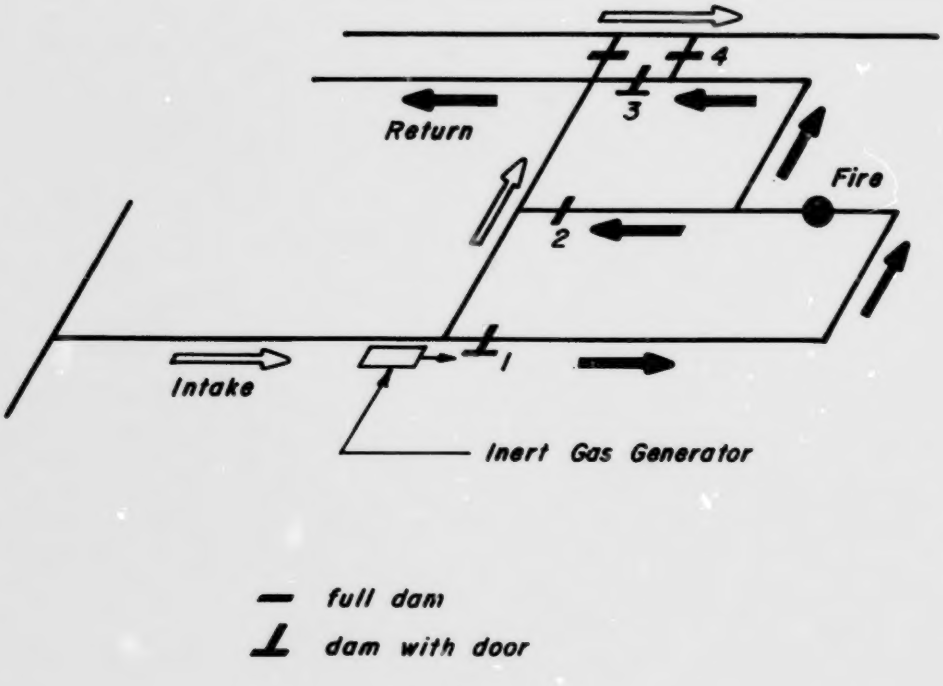


FIGURE 11-2

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

COOLING THE FIRE ZONE

BRZESZCZE MINE

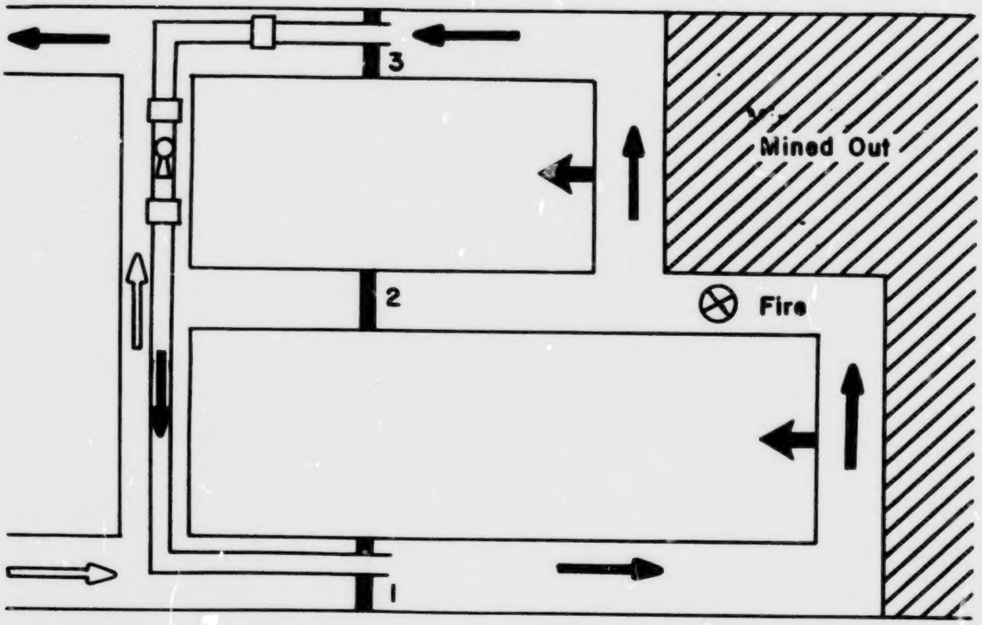
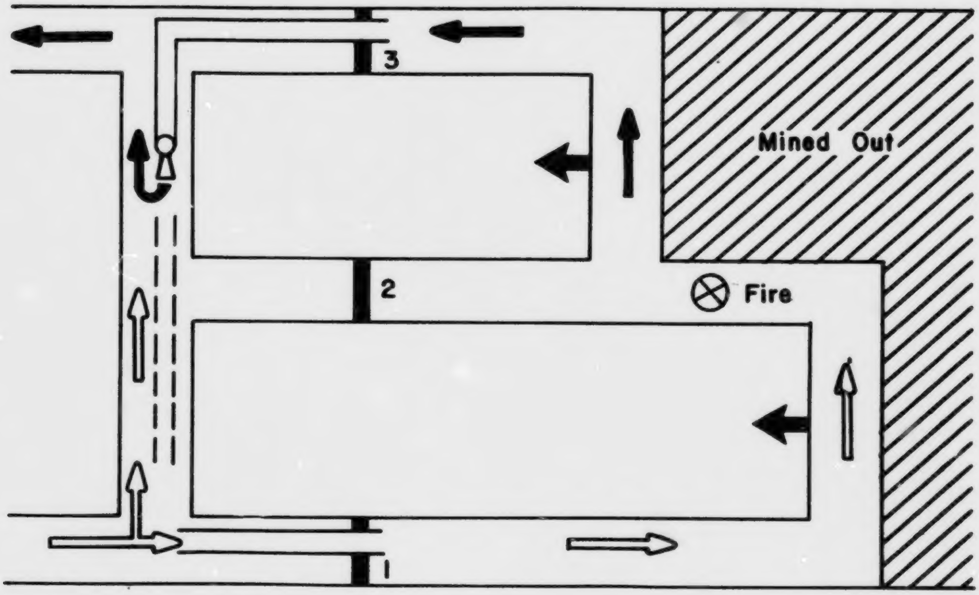


FIGURE 11-3
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

REVENTILATION OF THE SEALED ZONE
BRZESZCZE MINE



POLAND

STASZIC MINE

TYPE OF MINING	Longwall
DATE OF INCIDENT	1977
TYPE OF FIRE	Methane ignition on advancing longwall face
INERTING AGENT USED	Combustion products
DELIVERY SYSTEM	GAG inert gas generator installed underground
GAS DELIVERY RATE	Averaged 1000 m^3/min
TOTAL QUANTITY INERT GAS USED	3 300 000 m^3
SPECIAL FEATURES	First major action in which the GAG inert gas generator was used several times
REFERENCE	Kukuczka, A. - Personal communication. **Doswiadozenia Zakcji w Kopalni Doubrawa;** (Fighting a Fire at the Doubrawa Mine, Czechoslovakia).

1. BACKGROUND

A fire, caused by a methane ignition, occurred on October 5, 1977, on the #606 longwall face in the 501 seam of the Staszic Mine, Poland. It was decided to fight the fire using the GAG inert gas generator.

2. INERT GAS SUPPLY AND DISTRIBUTION

GAG inert gas generator was installed underground and GCP passed through fire stopping.

3. OPERATIONAL PROCEDURE

Figure 12-1 shows the layout of the mine in the area of the fire. The decision to use the GAG generator was taken at 2:30 am on October 5, 1977 some 4 hours after the fire started. The generator was brought from the nearby rescue station and installed underground and was ready for use within 13 hours.

The fire area whose volume was estimated to be 150 Mm^3 was partially sealed and the ventilation direction changed in the area to ensure fire gases exhausted away from active mining areas.

The generator was operated for 55 hours from October 5 - 10. It was found that substantial air leakage through the numerous seals, made reduction of the oxygen content difficult to achieve. The ventilation was adjusted to equalize the pressures acting on the seals, after which a low oxygen content was maintained in the fire zone. The total volume of inert gas fed into the fire area was 3 300 Mm^3.

To cool the fire zone and further reduce oxygen content, liquid nitrogen was pumped into the area through the seals at a rate of 8 m^3/min, but as transport was difficult, insufficient quantities were available to have any noticeable effect and consequently the operation was stopped.

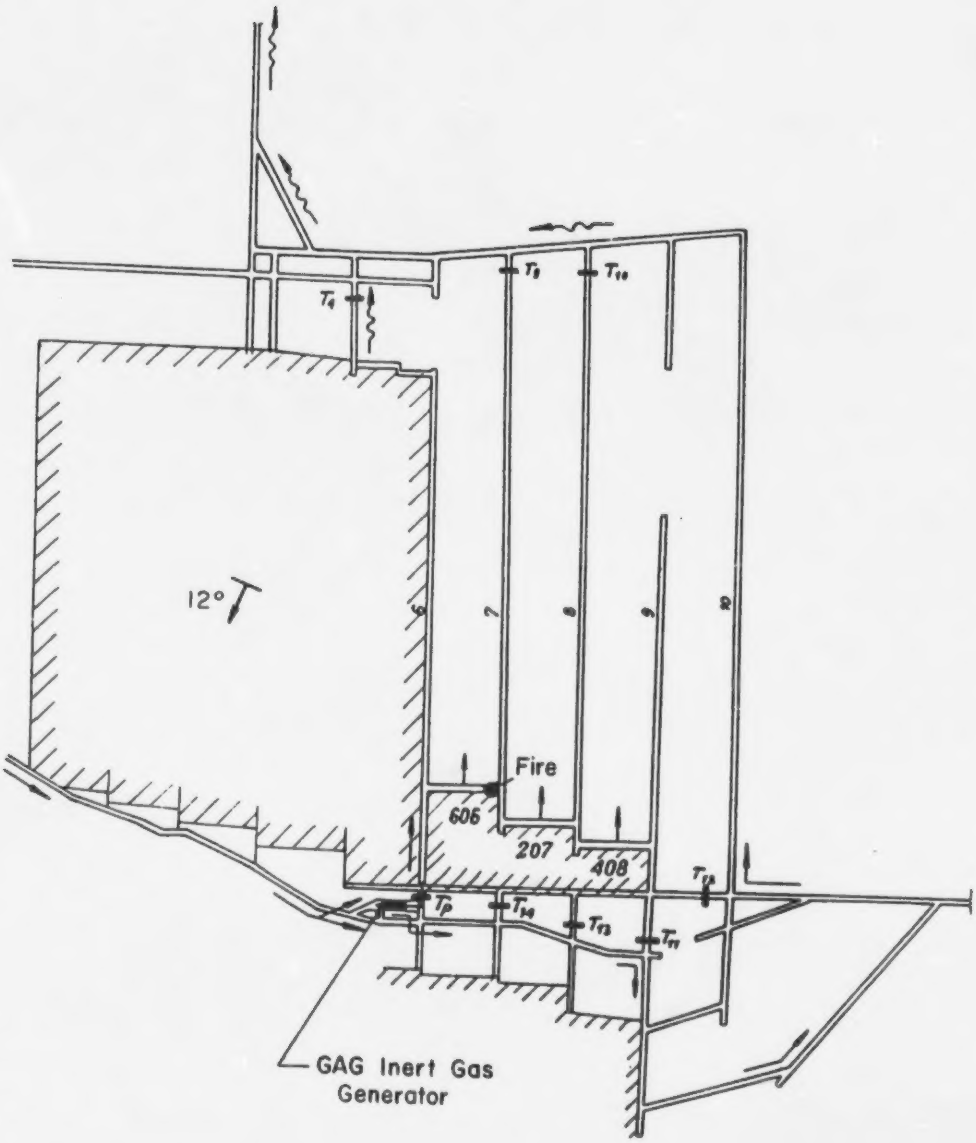
4. COMMENTS

This case illustrates the use of the GAG to reduce oxygen content in a large volume sealed area. The difficulties experienced in achieving low oxygen levels before the stoppings were balanced, highlights the importance of overall ventilation control during an inerting operation.

FIGURE 12-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

PLAN OF FIRE ZONE - STASZIC MINE



POLAND

ZABRZE MINE

TYPE OF MINING	Various
DATE OF INCIDENT	1976
TYPE OF FIRE	Spontaneous combustion between mine shafts
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Liquid nitrogen transported underground in special tanks and sprayed into fire zone
GAS DELIVERY RATE	
TOTAL QUANTITY INERT GAS USED	28 000 m^3
SPECIAL FEATURES	Use of liquid nitrogen to control mine fire
REFERENCE	Paczkowski, M and Kukuczka, A. 1978: **Zastosowanie Cielego Azotu do Gaszenia Pozarow w Kopalniach;** (Use of Nitrogen for Extinguishing Fires in Mines), Przeg. Gorn., v. 34/9, p. 375-380.

1. BACKGROUND

The Zabrze Mine in Poland works the 24 m thick 509/510 Seam by a variety of mining methods. On December 5, 1976 smoke was noticed in one of two intake shafts (shaft I and shaft II in Figure 13-1). Cracks had developed in the pillar between shaft collars. The pressure differential of approximately 800 N/m^2 between the shafts allowed a flow of air through the cracks, resulting in a spontaneous combustion heating.

Fire stoppings Tp2 and Tp3 (Figure 13-1) were closed and stopping Tp0 was erected and sealed to contain the fire zone. A small explosion damaged the stopping Tp0, so a stopping further outbye Tp1 was constructed. Gas analysis showed that the fire was increasing and threatening both the intake shafts, putting the total mine at risk. The decision was taken to utilize liquid nitrogen to control the fire. The same day the fire was discovered liquid nitrogen was considered so that a positive pressure could be built up in the fire sone to exclude further air, and to minimize the risk of further explosions.

2. NITROGEN SUPPLY AND DELIVERY

A stock of liquid nitrogen in tanks is maintained by the Mine Rescue Service, being delivered to the mine in 3 t - 15 t road tankers or 20-t - 50 t rail tankers. The liquid nitrogen was pumped into specially designed tank cars of 0.3 m^3 capacity, for transport underground. The nitrogen is forced from the tank by nitrogen gas from a cylinder at a pressure of 400 N/m^2. The liquid nitrogen flows through pipe passing through a fire stopping to spray nozzles. The mist of nitrogen produced is propagated through the fire area. The nitrogen droplets evaporate, taking some heat from the fire area and in expanding create a positive pressure.

3. OPERATIONAL PROCEDURE

On December 6, the day after the fire was discovered, the special tank cars were filled and delivered underground down shaft II, and pipe work and sprays set up at stopping Tp2. 50 nitrogen gas cylinders were stored at the shaft bottom.

Spraying of liquid nitrogen commenced at 5 pm on December 7, at a rate of 0.07 m^3/min and continued until December 11. Replacement of tanks caused breaks in the procedure. A total of 33 m^3 of liquid nitrogen was injected.

During the period, mine gas analyses at stopping Tp1 were taken and showed reducing values of CO, O_2, CH_4, CO_2 and H_2 with corresponding increase in N_2. After 17 m^3 nitrogen had been injected, the risk of explosive mixtures has passed.

On December 10, having brought the fire under control, the flap of stopping Tp3 was opened and stopping Tp1 dismantled. The mine ventilation was reversed, causing the fire gases to be cleared out via Shaft II. Rescue teams entered the area and reconstructed stopping Tp0. The effect was to reduce the fire area from 21 000 m^3 to 800 m^3. Liquid nitrogen was then sprayed through stopping Tp0 until December 12, when the fire was found to be completely extinguished.

4. COMMENTS

This case study illustrates the extinction of a fire in the shaft pillar area of a mine by liquid nitrogen, where conventional methods had failed.

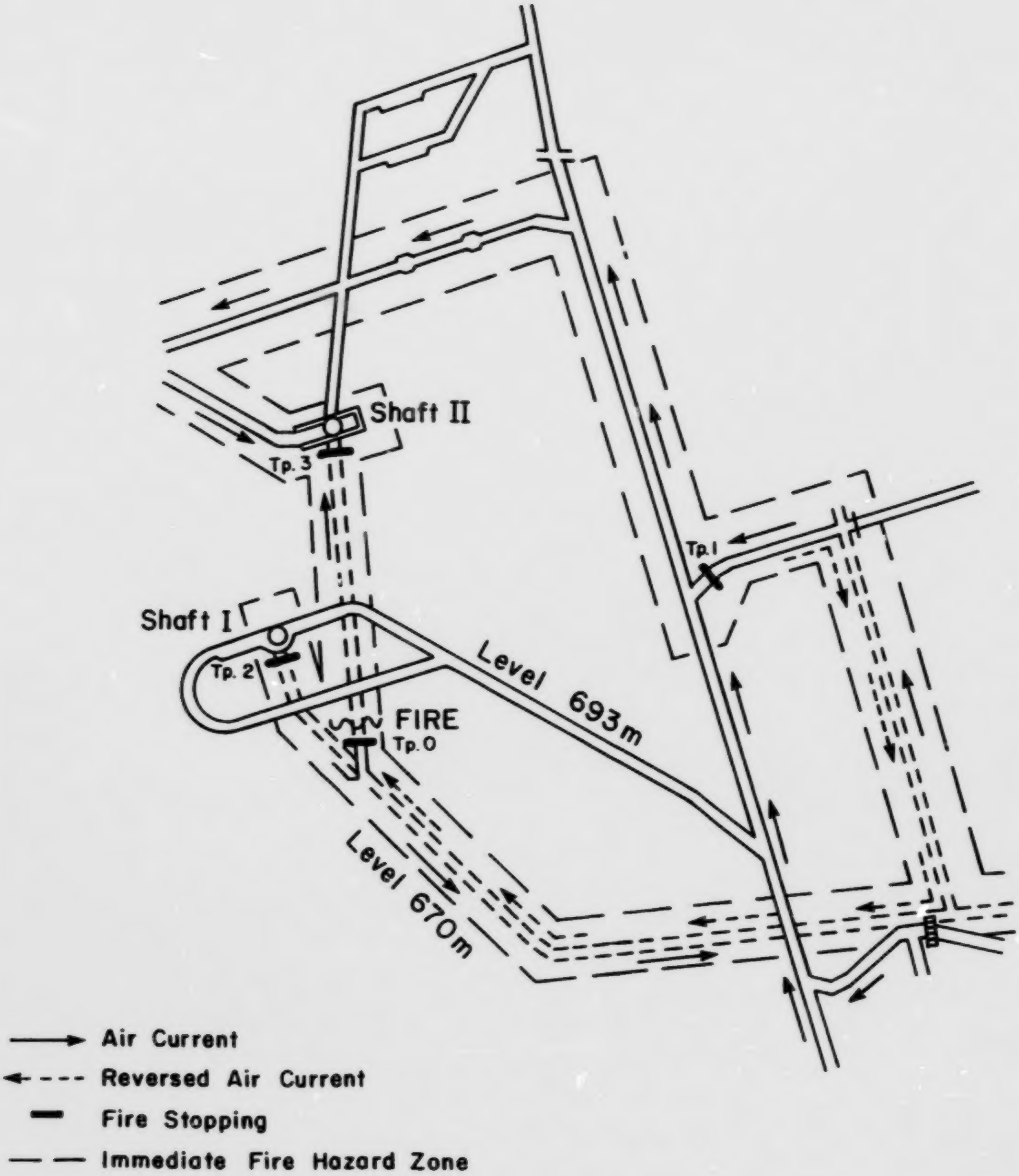
The closing of the fire area with stoppings created a serious explosion hazard which was neutralized by the addition of nitrogen. The application of liquid nitrogen was successful because:

- a fast response was possible
- it creased a positive pressure in the fire zone which restricted ingress of air
- it allowed the rescue teams to safely enter the area to erect a stopping
- it extinguished the fire.

This was the first use of liquid nitrogen underground in Poland to control a mine fire and it was recognized that improvements in technology and equipment were possible.

FIGURE 13-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

LOCATION OF THE MAIN ROADWAYS IN ZABRZE MINE



UNITED KINGDOM

DAW MILL 15's COLLIERY

TYPE OF MINING	Longwall
DATE OF INCIDENT	1980
TYPE OF FIRE	Heating in goaf of advancing longwall
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Liquid nitrogen vaporized on surface and delivered by pipeline to face
GAS DELIVERY RATE	5 - 40 m^3/min
TOTAL QUANTITY INERT GAS USED	3 593 368 m^3
SPECIAL FEATURES	Continuous use of nitrogen during face advance. First use of nitrogen in Britain to control heating on a longwall face.
REFERENCE	Harris, L. 1981: **The Use of Nitrogen to Control Spontaneous Combustion Heatings**; Min. Eng. (London), No. 237, 140, (Jun), p. 883-892.

1. BACKGROUND

In 15S District of the 7.3 m thick Warwickshire Thick Coal at Daw Mill Mine 3.5 m was extracted by a 230 m long advancing longwall face, equipped with ranging-drum shearers and 4-leg chock-shield supports.

The panel was in a known geologically disturbed area, but faulting on the face was worse than anticipated (Figure 14-1). The faulting slowed down the rate of advance of the face, which in turn led to spontaneous combustion problems.

The Warwickshire Thick Coal is particularly liable to spontaneous combustion, so preventative measures were taken on 15S to minimise the risk. Both roadways had pump packed gateside packs using the 'Thyssen System'. In addition, roadways were shuttered and backfilled with gypsum and a coating of gypsum sprayed on the outside of the shuttering. Special 'fire prevention' packs were built at the start of the face and the roadways on either side of the start-line were specially sealed.

In February 1980, a heating became evident in the goaf, which was thought to be along the line of faulting associated with the face start. The return air samples reached 40 ppm of CO before the heating was brought under control by continuing to advance the face to bury the heated area in the goaf, and by the use of a pressure chamber in the intake gate. Following a major roof fall near a fault, advance rates were slowed down in August and September 1980, due to the problems of grading through the faults and the need to dowel and forepole to prevent further falls. During this period the return airway immediately outbye the face was showing about 20 ppm of CO. From Monday, September 29, 1980 until Thursday October 2, 1980, the readings increased steadily to 70 ppm of CO, so a pressure chamber was introduced in the maingate and stopping sites prepared in both gates.

These preparatory sites were equipped with 1 m^2 section steel chambers with a view to re-entry at some future date. Pumping of hardstem from the maingate across the face start-line and across the fault, failed to reduce the heating. The decision was taken on October 3, 1980 to control the heating by nitrogen injection.

2. NITROGEN SUPPLY AND DISTRIBUTION

Liquid nitrogen, supplied by BOC-Nowsco Ltd was transported to site in 13 000 - 14 000 m^3 capacity tankers and converted to gas by diesel-fired vaporizer. The equipment was capable of delivering 118 m^3/min at 35 Bar although ones 30 m^3/min at 10 Bar was required.

The nitrogen was delivered through existing pipework to the face. The 2 738 m length of pipework had 100 mm bore in the shaft and 75 mm bore elsewhere.

3. OPERATIONAL PROCEDURE

The nitrogen was introduced into the waste behind the first-line chock on October 3, 1980, followed by injection through the gateside pack about 40 m from the face by No.1 pipe. Within 6 hours, CO readings stabilized around 80 ppm from a peak of 112 ppm. On October 4, 1980 further pipes were inserted into the waste behind the pack at varying intervals from the face. The pressure chamber was taken off to allow No.2 pipe to be injected and No.1 pipe stopped. This had the effect of purging the waste and the CO in the return peaked at 116 ppm shortly afterwards. Further pipes on a manifold arrangement were installed and 30 m^3/min of nitrogen injected through pipes 2 - 6 on October 5, 1980. CO levels dropped within 24 hours from 100 ppm to 25 ppm. Airflow had to be increased in the district to ensure O_2 levels were maintained above the regulatory 19% and methane levels below 0.5%.

By late October 5, 1980 face operations were able to resume. Over the following weeks nitrogen injection was progressively reduced from 30 m^3/min down to 10 m^3/min, although it had to be increased occasionally to arrest a rise in CO levels.

Following a further major rooffall in early December, a second incidence of spontaneous combustion was indicated by increased CO readings. The nitrogen was increased to 40 m^3/min, injected through pipes 2, 3 and 5 and through two pipes installed on the face-line. These latter pipes were in 2 m screwed sections, the leading pipe being perforated and having a spade end. They were driven about 7 m into the waste. With the increased nitrogen flow, and reduced airflow around the district, auxiliary ventilation was necessary in the return gate so the O_2 content was maintained (Figure 13-2). Hardstem pumping from the return gate and from the face-line was also used to tackle the problem. By December 20, CO levels had again fallen to below 20 ppm.

Injection of nitrogen continued while the face continued to advance to its working limit.

Costs of the nitrogen injection up to December 31, 1980 amounted to \$950 000 at which time 3 015 000 m^3 of nitrogen had been used. By allowing the face to continue \$7 million worth of equipment and 800 000 t of coal reserves were saved.

4. COMMENTS

This example demonstrates how the use of nitrogen prevented a district from being sealed off, and the face to continue operation.

FIGURE 14-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

DAW MILL MINE - 15S DISTRICT

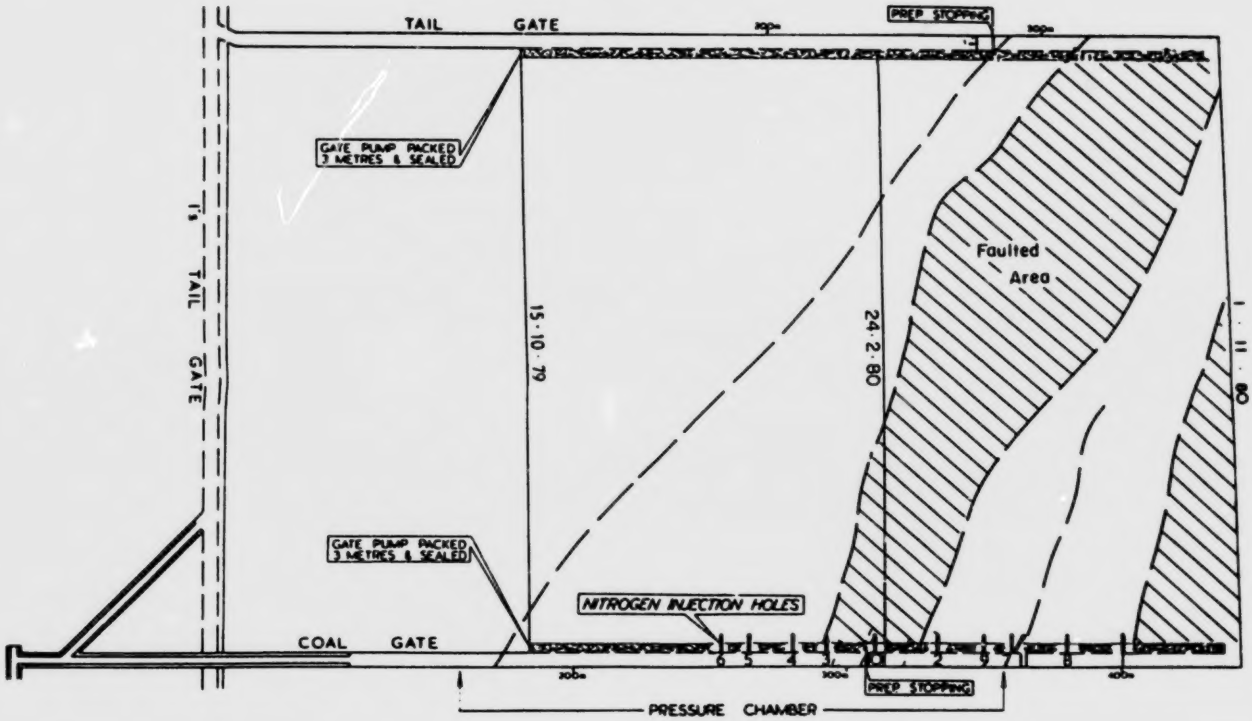
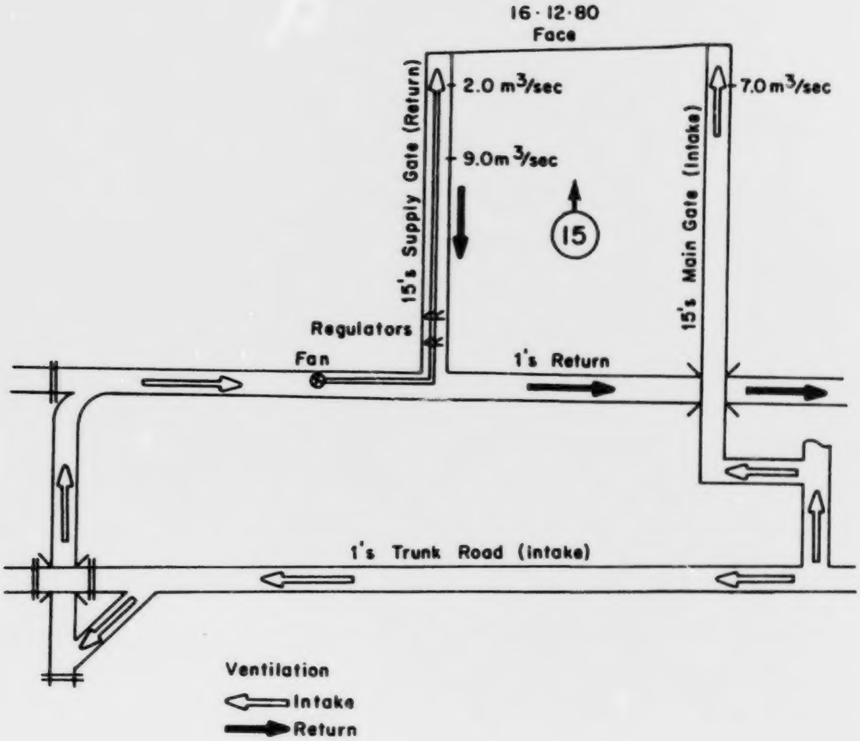


FIGURE 14-2

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

AIR QUANTITY CHANGES WHERE NITROGEN FLOW INCREASED TO 40 m^3/min



UNITED KINGDOM

DAW MILL 65's MINE

TYPE OF MINING	Longwall
DATE OF INCIDENT	1983
TYPE OF FIRE	Heating of goaf of advancing longwall
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Liquid nitrogen vaporized on surface and delivered by pipeline to district
GAS DELIVERY RATE	5 - 60 m^3/min.
TOTAL QUANTITY INERT GAS USED	N/A
SPECIAL FEATURES	Heating reactivated after inerted area was ventilated
REFERENCE	Lord, S.B. 1983, **Some Aspects of Spontaineous Combustion Control**; Inst. Min. Eng. (London), South Staffordshire and South Midlands Branch, November 14, 50 p., (London).

1. BACKGROUND

Daw Mill 65's was a 220 m longwall face advancing in the Two Yard Section of the Warwickshire thick coal. The face was equipped with heavy duty supports for 365 m extraction. The gates were pump packed. The normal rate of advance was 8 m/week.

After advancing 770 m the face ran into a geologic disturbance and came almost to a standstill. From November 1982 to August 1983 only 30 m further forward progress was made.

In mid-March 1983 the CO level at the control point (Figure 15-1) began to rise from the district norm, it reached 25 ppm within a week, and then there was a sharp rise to 125 ppm.

At this time, influenced by the success on 15's district (Case Study #13), it was decided to introduce nitrogen.

2. NITROGEN SUPPLY AND DELIVERY

Liquid nitrogen, supplied by BOC-Nowsco Ltd. was transported to site in 13 000 - 14 000 m^3 capacity tankers and converted to gas by diesel-fired vaporizer. The equipment was capable of delivering up to 118 m^3/min of nitrogen at 35 Bar.

The nitrogen was delivered through existing pipework to the panel then through boreholes driven through the gateside packs as indicated on Figure 15-2.

3. OPERATIONAL PROCEDURE

Nitrogen was injected at a total flow rate of 25 m^3/min initially into four holes drilled at the 735, 848, 817 and 786 m positions. The marks were measured from the outbye end of the return gate, the face being at 910. This had the effect of reversing the rising CO trend. The methane percentage rose to almost 1.25%, as a result of a combination of it being forced from the goaf by the nitrogen and the expansion effect of a falling barometer.

Over the following three days the nitrogen was lifted progressively to 40 m^3/min, the CO level fell to 75 ppm, the methane to 0.5% and the oxygen dropped to 19%. In this time the hole at 735 had to be abandoned as it became blocked with sealant from adjacent injection holes. Holes at 860 and 700 were commissioned.

On the following day, March 26, 1983, the downward trend of the CO and methane samples reversed, the CO rose to 175 ppm and the methane again approached 1.35%, possibly due to a half hour interruption which occurred in the nitrogen supply. At this time the nitrogen injection rate was increased to 50 m^3/min.

Over the next 24 hours there was a rapid drop in both CO and CH_4, to 125 ppm and 0.5% respectively. The oxygen also fell to 18% as a result of the increase in nitrogen injection rate. On the next day, March 28, there was an extremely rapid rise in the CO concentration which reached a peak of 350 ppm. This was very shortlived and it fell back to 225 ppm late in the day, and to 125 ppm over the following 24 hours. It was significant that this sudden rise and fall in the carbon monoxide concentration was accompanied by a fall in oxygen to 16% and a rise in methane from 0.5% to 0.9%.

From this time onwards the fall in CO concentration continued, the nitrogen injection rate was increased to 60 m^3/min and the oxygen in the return gate fell to a minimum of 15%. Over the next three days the oxygen stabilised at 17%, the methane at 0.5% and the CO level fell to 15 ppm.

Throughout this nine day period a total of 19 holes were used for nitrogen injection, of which three were on the face. Holes at 860, 786 and 700 were on line for most of the time. Continuous pumping of sealant both in the maingate and to a more limited extent on the face took place throughout, although work on the face was limited by the position of the 19% oxygen fringe.

From April 1 onwards until the August 5 work was resumed on the face with the intention of working it through the disturbance. Conditions were such that little progress could be made and on August 5 the sample at the control point began to rise again.

The samples indicated a rapidly deteriorating situation over the following two days and a decision was made to temporarily seal the district off. This was effected on August 7, 1983. It was done with the intention of re-opening the district at a later date, with the object of salvaging part of the face and gateroad equipment.

To facilitate the re-entry the district stoppings were constructed with access tubes. The nitrogen range was left coupled to holes in the intake gate and the total flow rate was reduced to approximately 5 m^3/min. It was considered that leaving the nitrogen injection range operational during the sealed off period would be beneficial for three main reasons. It would considerably help to cool the strata in the vicinity of the heating and thereby reduce the likelihood of re-activity on re-opening. It would ensure that any residual oxygen in the district after closure would be replaced by nitrogen. Finally, and most important, by continually displacing the methane make, it would eliminate the possibility of an explosive mixture being formed on dilution with oxygen during the re-opening.

The district was sealed off from August 7 until September 16. During this period the pressure between the intake and return gates was balanced by means of a pressure chamber at the return stopping. Both intake and return stopping pressures were monitored at 50 Pa and 55 Pa respectively. This pressure being generated across the stoppings by the outflow of nitrogen. Samples taken from behind the stoppings rapidly stabilised and typically contained 5% methane 1% oxygen and 15 ppm carbon monoxide.

The district was re-opened on September 16. With the colliery clear, and the electricity in the return isolated, the return stopping was over pressurized by means of the chamber, allowing a rescue team to safely remove the access tube filling. When this was completed the chamber fan was switched off and the regulator doors removed. The intake stopping access tube was then cleared and an air quantity of 3 m^3/s was measured as flowing through the district. When samples at the remote sampling station in the return slit indicated that the plug had passed, a district inspection was made. The district was found to be cool and no signs of the heating could be seen. The samples taken at the district control point contained 11 ppm of CO on re-opening. Unfortunately to CO content then rose continuously at an increasing rate and the district had to be re-sealed 48 hours later at which time the CO concentration had reached 57 ppm. The nitrogen injection was discontinued.

A chart showing the sequence of holes used for nitrogen injection is shown as Figure 15-3.

4. COMMENTS

This example demonstrates a case where a goaf heating that had been controlled with nitrogen in a sealed area was reactivated when the seals were opened and ventilation was restored.

FIGURE 15-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

DAW MILL MINE 65's HEATING

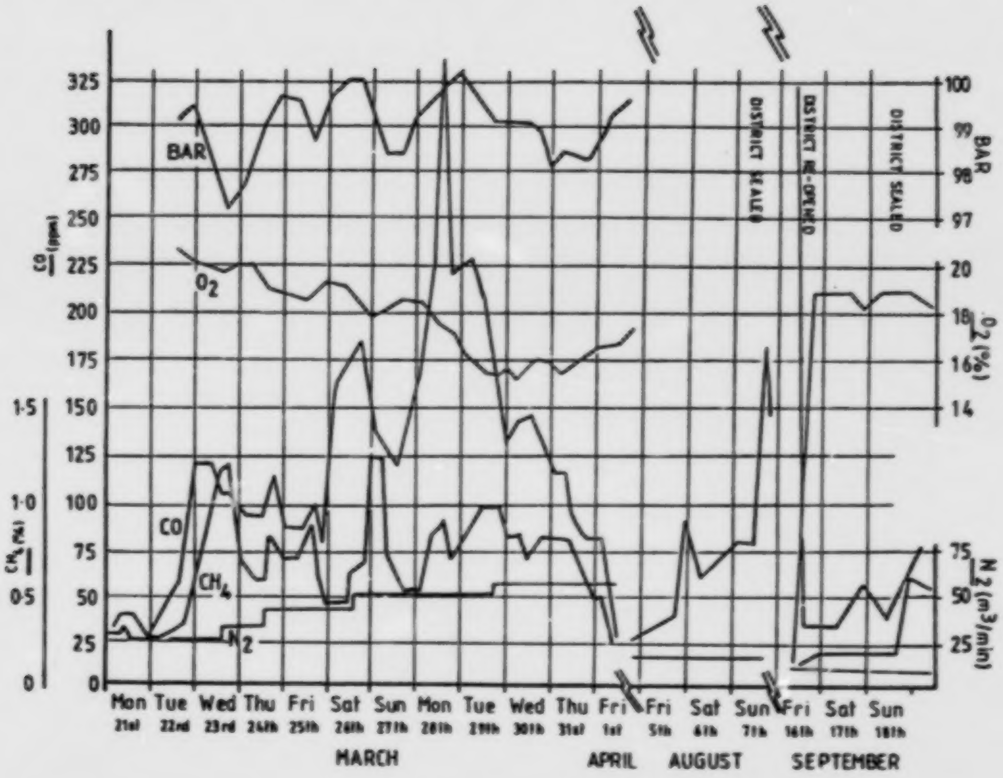


FIGURE 15-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

DAW MILL MINE BOREHOLES AT 65's HEATING

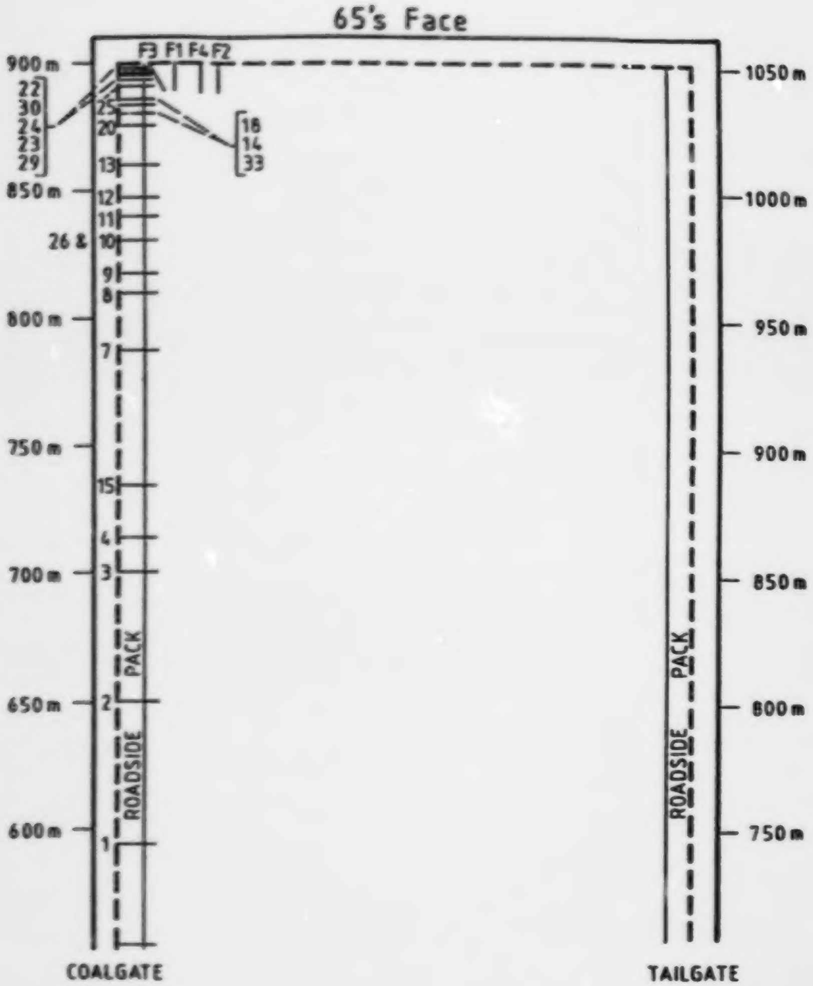
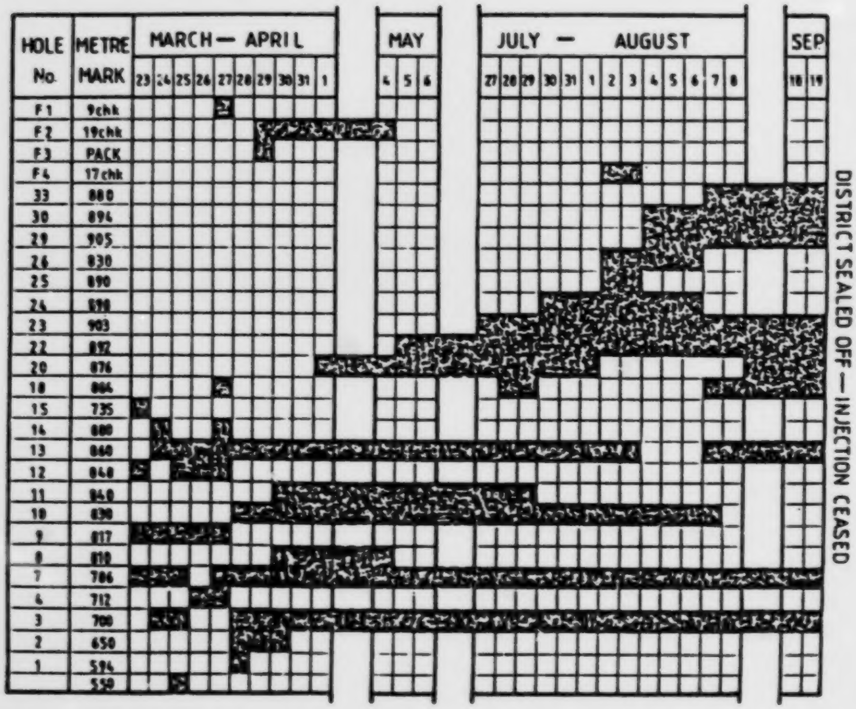


FIGURE 15-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

DAW MILL MINE

NITROGEN INJECTION - 65's FACE 1983



Case Study #16

UNITED KINGDOM

FRYSTON MINE

TYPE OF MINING	Longwall
DATE OF INCIDENT	1981
TYPE OF FIRE	Heating in goaf of retreating longwall
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Liquid nitrogen vaporized on surface and delivered through surface borehole into goaf
GAS DELIVERY RATE	2.5 - 30 m^3/min
TOTAL QUANTITY INERT GAS USED	765 180 m^3
SPECIAL FEATURES	Use of surface borehole to deliver nitrogen.
REFERENCE	Wastall, E.R. and Walker, G. 1983: **The Use of Nitrogen at Fryston Colliery**; Min.Eng. (London) (Jul), p. 27-36.

1. BACKGROUND

Panel 74S in the Beeston Seam in Fryston Mine was worked by a 250 m long retreating longwall face that was stopped by washout conditions (Figure 16-1).

A new face (76S) equipped with heavy duty supports and face equipment was developed adjacent to 74S, and was due to start production on March 16, 1981. On March 12, 1981, when only gateroad equipment was left to withdraw from 74S a slight increase was noted in the tube bundle CO readings in 74S tailgate. Supplementary sampling by full analysis capsule sampling and Draeger readings indicated no cause for alarm, so the decision was taken to withdraw the remaining equipment and seal off the district the following week. On Sunday March 15, 1981, however, dense smoke and high temperatures indicated a goaf heating in 74S maingate inbye of the face with CO readings rising to 3000 ppm. Rescue teams sealed off secondary stoppings in the east Intake and Return by March 17, 1981, allowing normal deployment elsewhere in the mine. A detailed sampling program was established and a mobile laboratory with full analysis facility set up at the mine.

Unless a safe method of building the primary seals in 74S could be achieved, the $6.5 million worth of equipment on 76S and access to 3 million tonnes of reserves would be lost. It was decided to inject nitrogen into the area of the heating, to allow safe re-entry to the district. The nitrogen was pumped through a surface borehole because:

- the pumping distance otherwise would have been 8 km
- it would be uncertain what happened at the fire site if nitrogen were pumped in behind the secondary stoppings
- by injecting nitrogen to the inbye end of 74S the only area which would contain residual methane would be 76S.

2. NITROGEN SUPPLY AND DISTRIBUTION

Liquid nitrogen, supplied by BOC-NOWSCO Ltd. was transported to site in 14 000 m^3 capacity tankers and converted to gas by diesel-fired vaporizer. The nitrogen at 15°C was pumped down the borehole at a surface pressure of 3445 kN/m^2 at a rate of up to 40 m^3/min.

The borehole was drilled to intersect the waste 22 m behind the face line and 22 m in from the maingate side, in order to avoid striking any residual steel and to permit deviation tolerances during drilling. Drilling was through a blow-out preventer, and the hole of 120 mm finished diameter was completed by April 14, 1981, 18 days after the decision to drill was taken.

Nitrogen was pumped down the drill rods to ensure its arrival at the seam horizon.

3. OPERATIONAL PROCEDURE

Table 16-1 shows the times, rates and quantities of nitrogen injected.

An extinctive atmosphere was achieved behind both secondary stoppings during the first weekend that N_2 was injected, April 4 and 5, 1981, and it was concluded that active combustion had ceased. The effect of injection on the mine atmosphere in the east Return and in 74S tailgate is shown in Figures 16-2 and 16-3 respectively. They show that while an inert atmosphere was maintained behind the stoppings, the rest of the mine was safe for normal working. The rapid response time to changes in rate of injection was noted.

Recovery of the district depended on placing and sealing primary stoppings in the 74S gateroads. After degassing, auxiliary ventilation was established at the primary stopping sites, with nitrogen injected to prevent oxygen reaching the source of the heating and consequent risk of explosion. The stoppings consisted of two ageite block walls, 5 m apart, into which hardstop was pumped from outbye the secondary stopping. After the primary stoppings were sealed, it was possible to remove the secondary stoppings and degass 76S district. The 76S face was then able to start production.

Costs of the recovery operation were approximately $0.6 million, of which $0.3 million were for the nitrogen. This saved $6.5 million of equipment and 3.5 Mt of saleable coal.

4. COMMENTS

This example illustrates the use of a borehole to inject nitrogen into a goaf heating, allowing oxygen to be displaced from a sealed area. The maintenance of an inert atmosphere under a positive pressure in the heated area allows the sealed area to be re-entered quickly and safely. Auxiliary fan ventilation can be used without the danger of air being passed over the fire area.

It was shown that low flowrates of nitrogen are required when it is injected near the source of the heating.

TABLE 16-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

NITROGEN INJECTION

Date	Time		Date	Time	Rate cu.m/min	Amount m^3
4.4.81	06.00	to	5.4.81	21.00	30 to 50	64 000
11.4.81	12.00	to	12.4.81	21.00	30	60 000
22.4.81	11.50	to	25.4.81	03.00	5	17 695
25.4.81	03.00	to	26.4.81	24.00	30	80 750
27.4.81	01.00	to	13.5.81	10.00	10	236 235
20.5.81	10.30	to	22.5.81	15.45	30	93 600
28.7.81	08.45	to	30.7.81	15.50	30	99 000
30.7.81	16.45	to	1.8.81	18.40	5	15 000
1.8.81	18.40	to	1.8.81	19.20	2.5	100
1.8.81	19.20	to	2.8.81	06.00	5	3 200
2.8.81	06.04	to	2.8.81	12.18	10	3 750
2.8.81	12.18	to	2.8.81	14.35	15	2 025
2.8.81	14.35	to	2.8.81	15.55	5	375
2.8.81	15.55	to	6.8.81	09.55	15	81 800
6.8.81	09.55	to	7.8.81	12.30	5	7 650
Grand Total						765 180

FIGURE 16-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FRYSTON COLLIERY - BEESTON SEAM VENTILATION FEATURES

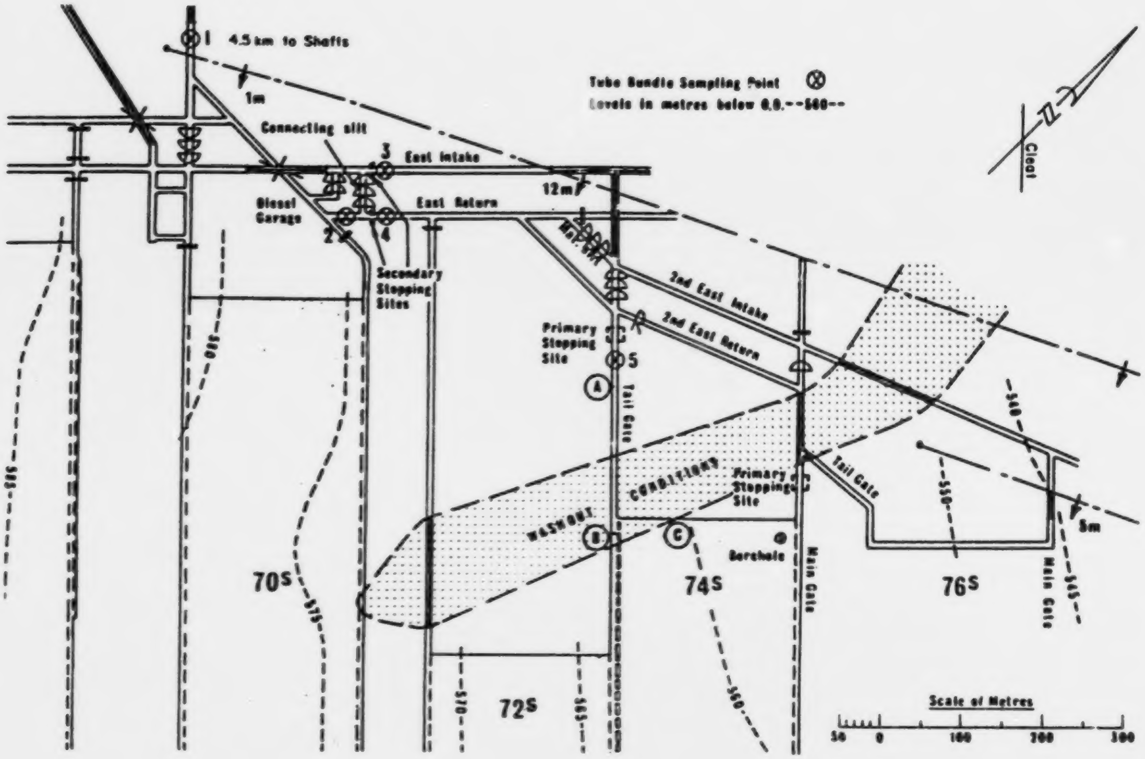


FIGURE 16-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

GRAPH OF ATMOSPHERE ANALYSES AT POINT 2 IN THE EAST RETURN

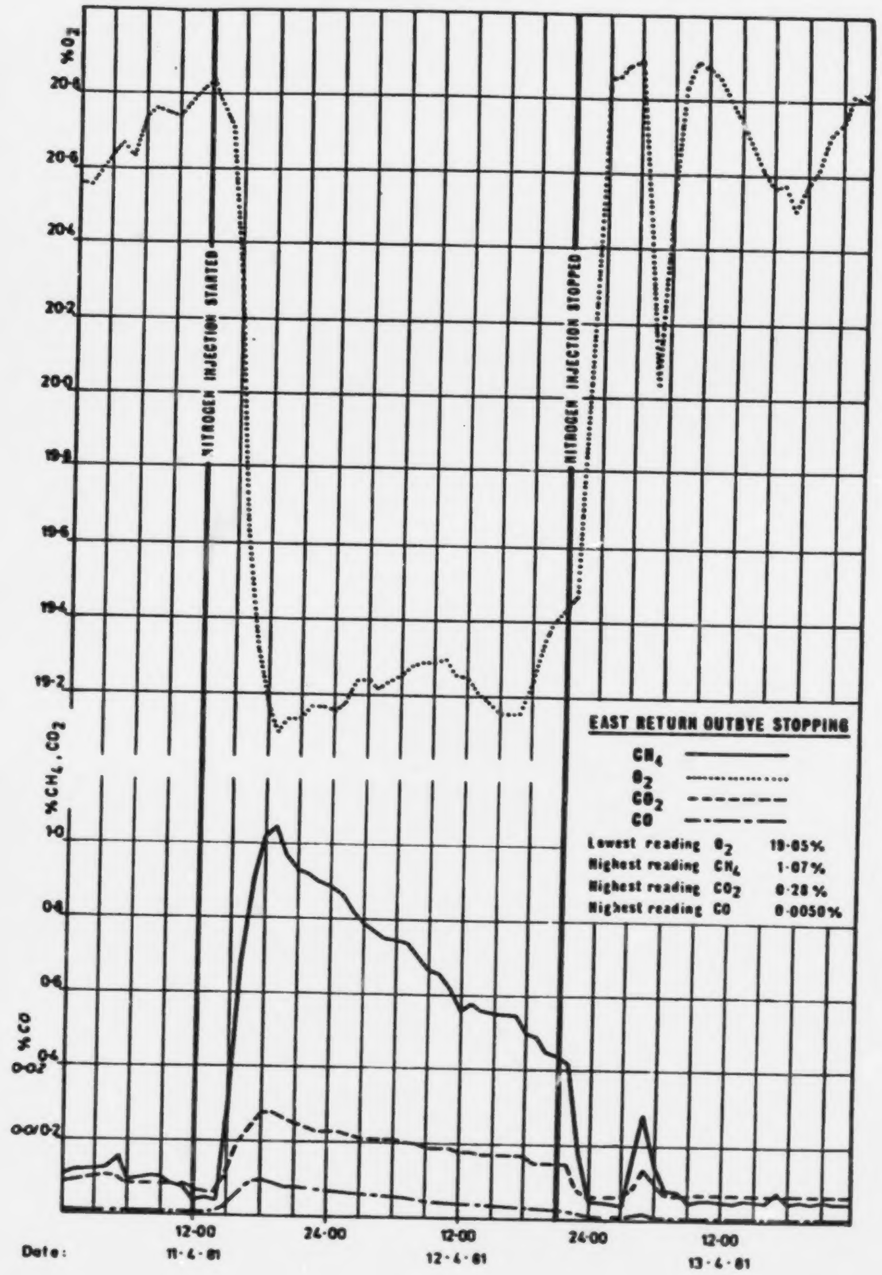
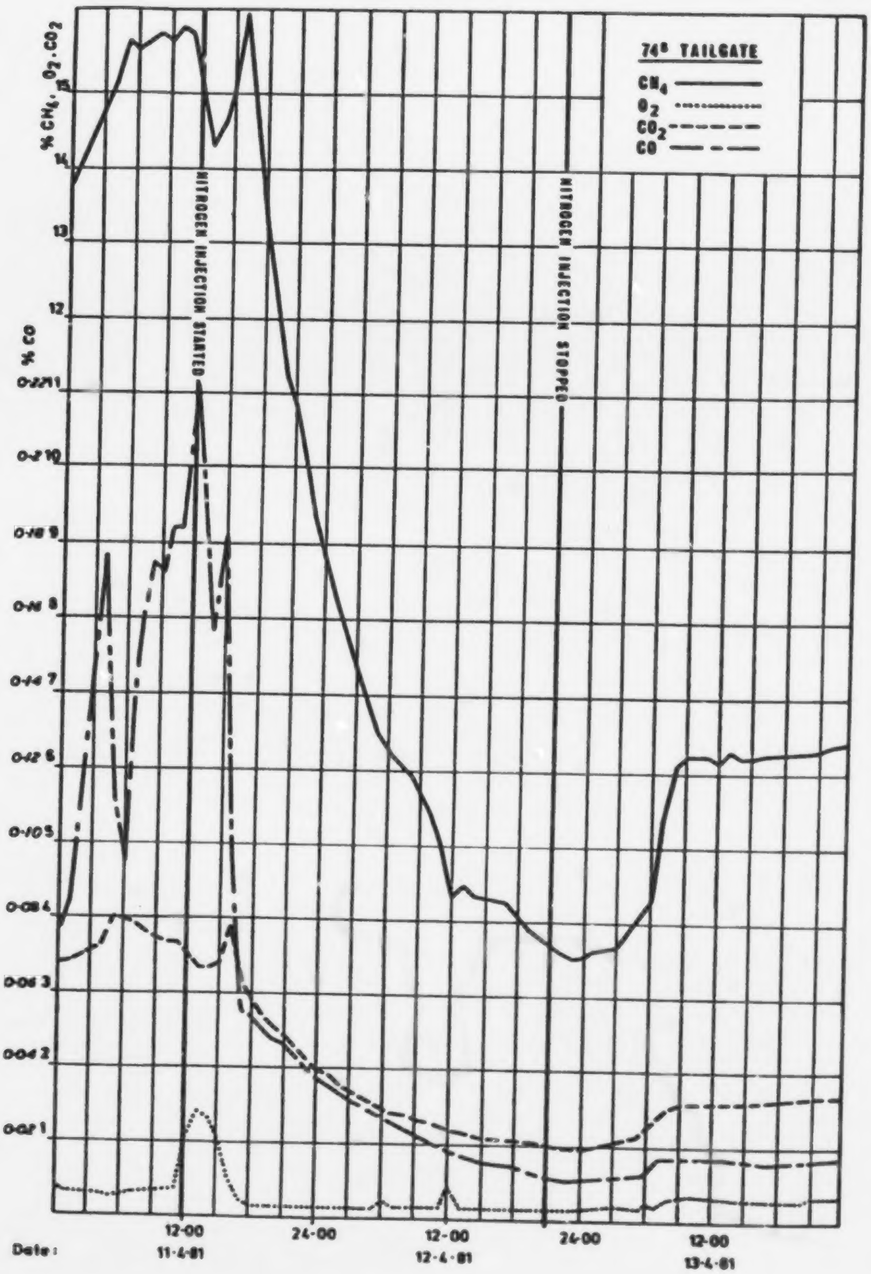


FIGURE 16-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

RESULTS OF OXYGEN DISPLACEMENT AT 74S TAILGATE



Case Study #17

U.S.A.

INLAND #1 MINE

TYPE OF MINING	Room-and-pillar
DATE OF INCIDENT	1980
TYPE OF FIRE	Spontaneous combustion in caved area
INERTING AGENT USED	Carbon Dioxide
DELIVERY SYSTEM	Liquid carbon dioxide injected through surface boreholes
GAS DELIVERY RATE	Variable
TOTAL QUANTITY INERT GAS USED	357 t CO_2 (liq) (192 000 m^3 gaseous CO_2)
SPECIAL FEATURES	
REFERENCE	Wolfe, M. 1980: **Report of Investigation - Spontaneous Combustion (Gob) Fire, Inland #1 Mine;** US Department of Labour Mine Safety and Health Administration (MSHA).

1. BACKGROUND

The Inland #1 Mine of the Inland Steel Coal Company, located in Jefferson County, Illinois. The 1.8 m thick Illinois No. 6 Seam is worked by conventional room-and-pillar methods.

On Monday, July 21, 1980 at abour 3:10 pm, company officials informed the Subdistrict office of Mine Safety and Health Administration in Benton, Illinois, that carbon monoxide was detected coming from the 16 right bleeder entry, indicating a (gob) mine fire existed (Figure 17-1). All persons were withdrawn from the mine.

A check of 16 right bleeder entry with a carbon monoxide detector indicated the presence of carbon monoxide ranging from 10 ppm to 250 ppm. Air samples were collected and the results of analyses showed a range of 27 ppm to 169 ppm. There was no odour, haze, smoke, heat or condensation detected in the 16 right bleeder. It was decided that the safest, most efficient, and quickest way to control the fire would be to seal the area, then add liquid CO_2 to inert the atmosphere behind the seals.

2. CARBON DIOXIDE SUPPLY AND DISTRIBUTION

Liquified carbon dioxide was supplied in tankers and pumped down cased boreholes drilled from surface.

3. OPERATIONAL PROCEDURE

The sealing of the fire area consisted of the closing of two four-by-four openings at each end of the bleeder system (one at 16 right and one at 17 right). The sealing operation was completed at 11:15 pm on July 21, 1980. Two inch sampling pipes were installed in 16 right and 17 right, for the purpose of collecting samples of the atmosphere behind the seals.

Boreholes were drilled to penetrate the 16 right and 17 right. After the completion of the boreholes July 23, 1980, 24 t of liquified carbon dioxide were injected at the 17 right borehole. On July 24, 1980, an additional 17 t were injected at 17 right and 51 t in 16 right. After these injections, the integrity of the seals were checked and samples collected from the sealed area. The samples showed 1.05% methane, 18.72% oxygen and 131 ppm carbon monoxide. As a result of these samples, company officials decided to inject additional liquified carbon dioxide into the area, 17 t in 17 right and 17 t in 16 right. On July 25, 1980, samples were collected and their analyses showed 16 right seal to be 1.97% methane, 10.54% oxygen and 165 ppm of carbon monoxide; 17 right seal showed 3.04% methane, 12.44% oxygen and 137 ppm of carbon monoxide. Resumption of operations was then permitted.

The atmosphere behind the seals was monitored continuously and samples were collected each shift. As a precautionary measure, company officials elected to continue injecting liquified carbon dioxide into the sealed area. All persons were removed from the mine during injection of the liquified carbon dioxide. A total of 357 t of liquified carbon dioxide was injected into the sealed area. The results of analyses collected on September 9, 1980, showed 28.45% methane, 4.07% oxygen and 46 ppm of carbon monoxide at 17 right seal, and 16.89% methane, 3.23% oxygen and 179 ppm of carbon monoxide at 16 right seal.

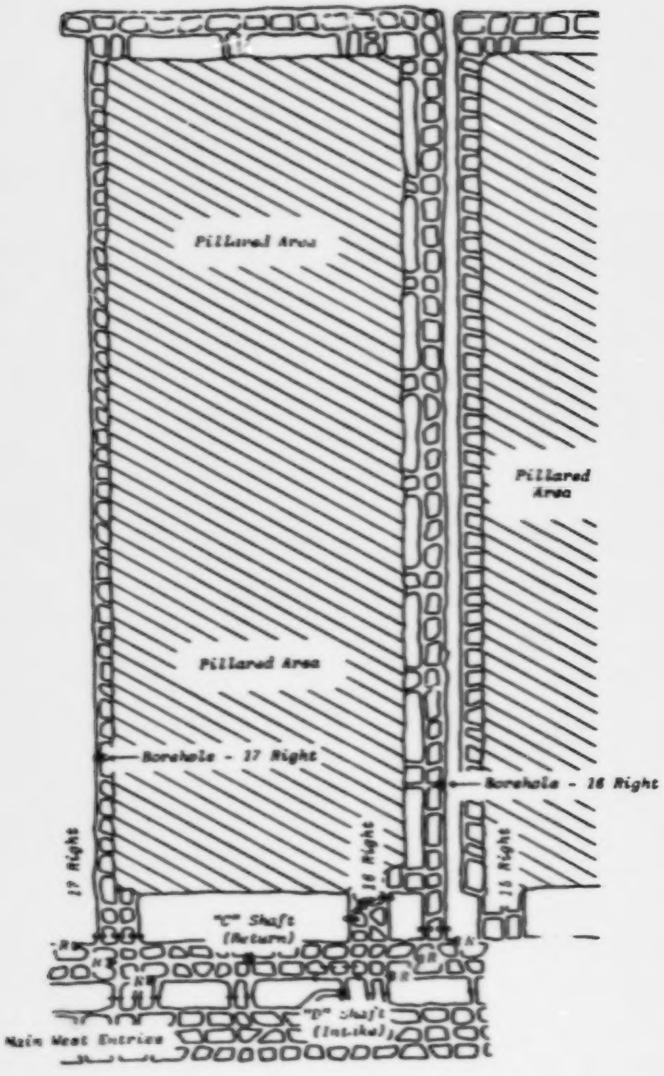
4. COMMENTS

This case study demonstrates how a fire in a caved area can be isolated by use of seals, and how liquid carbon dioxide can be injected via surface boreholes into the sealed area to inert the atmosphere and control the fire.

FIGURE 17-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

LAYOUT OF FIRE AREA - INLAND #1 MINE



Case Study #18

U.S.A.

JOANNE MINE

TYPE OF MINING	Room-and-Pillar
DATE OF INCIDENT	1974
TYPE OF FIRE	Electrically initiated fire in roadway
INERTING AGENT USED	Nitrogen
DELIVERY SYSTEM	Liquid nitrogen used with fly ash and foam for remote sealing through boreholes
GAS DELIVERY RATE	Variable
TOTAL QUANTITY NITROGEN USED	Not known
SPECIAL FEATURES	Use of remote sealing through boreholes using inert gas
REFERENCE	King, Roger L. **Remote Sealing System for Extinguishing Coal Mine Fires;** USBM publication.

1. BACKGROUND

The Joanne Mine of the Eastern Associated Coal Corporation in Rachel, West Virginia, is worked by room-and-pillar methods.

In January 1974, a derailment and subsequent short circuit resulted in a fire in the underground working (Figure 18-1). The short circuit caused a fan stoppage and made it necessary to evacuate the mine. The decision to attempt remote sealing was taken some 51 days later.

2. NITROGEN SUPPLY AND DISTRIBUTION

Liquid nitrogen was supplied in tankers and used as inerting medium for the injection of fly ash seals. Because of supply and delivery problems it was not always possible to maintain an inert atmosphere in the sealed area for extended periods during the recovery phase.

3. OPERATIONAL PROCEDURE

The remote sealing system using fly ash was developed by the USBM. The main components of the system are fly ash supply trucks, a diesel driven blower, liquid nitrogen supply tankers to supply an inerting medium and a plumbing network with instrumentation to monitor flow rates and pressure. Fluidized fly ash is fed from 10 t supply trucks to an eductor where it is mixed with nitrogen-diluted air (less then 8% O_2). The mixture is pneumatically transported through a 200 mm diameter fibreglass pipe to the borehole by a high capacity blower at rates of up to 70 m^3/min. The fly ash fills the mine passageway to create an effective seal and can be topped with inert foam. The foam topping system provides an expanding material that is air-tight and water-tight. The foam uses Freon 12 as the inert gas (Figure 18-2).

At Joanne Mine the remote sealing was made from two surface sites over 760 m apart, and located on opposite sides of a hill above the mine. The sites were selected because of their proximity to existing roadways, sufficient distance from the fire zone, adequate distance from existing gas wells, and would avoid burying major pieces of mining machinery. Six seals were constructed, each requiring two adjacent boreholes to accommodate accoustic seal checkers. The boreholes were up to 183 m deep. Three seals were 95% fly ash topped with foam, while the other three were 100% fly ash seals. Between 150 t and 500 t of fly ash was required for each seal and where used, 1 t of foam per seal. The sealing operation required at least 18 400 m^3 nitrogen per seal for deploying the fly ash.

Within a few days of completion of the seals it was possible to reventilate the mine and recommence operations.

The gas composition of the mine air at a monitoring point during the incident is shown on Figure 18-3.

1. COMMENTS

This case study shows how the use of inert gases (nitrogen and freon) with remote sealing through boreholes to extinguish fires with minimum danger to personnel.

FIGURE 18-1

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

MAP OF UNDERGROUND SECTION OF JOANNE MINE SHOWING FIRE ZONE AND FLY ASH SEALS

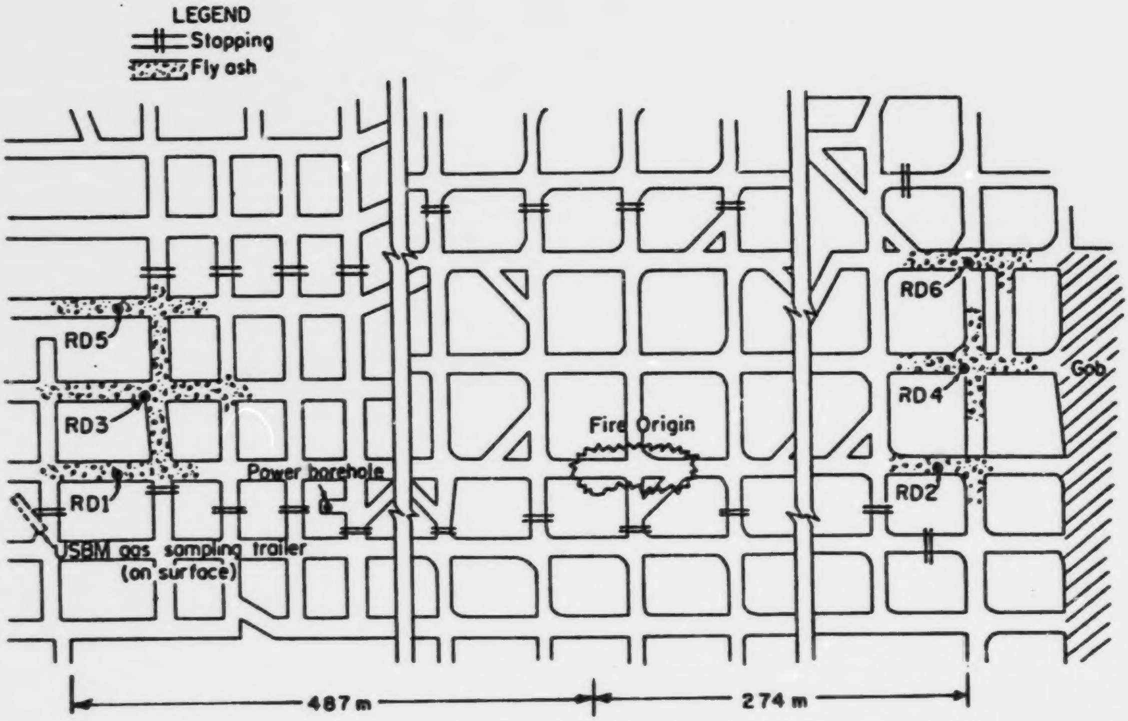


FIGURE 18-2

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FROTH FOAM CONDITIONING, METERING AND MIXING SYSTEMS

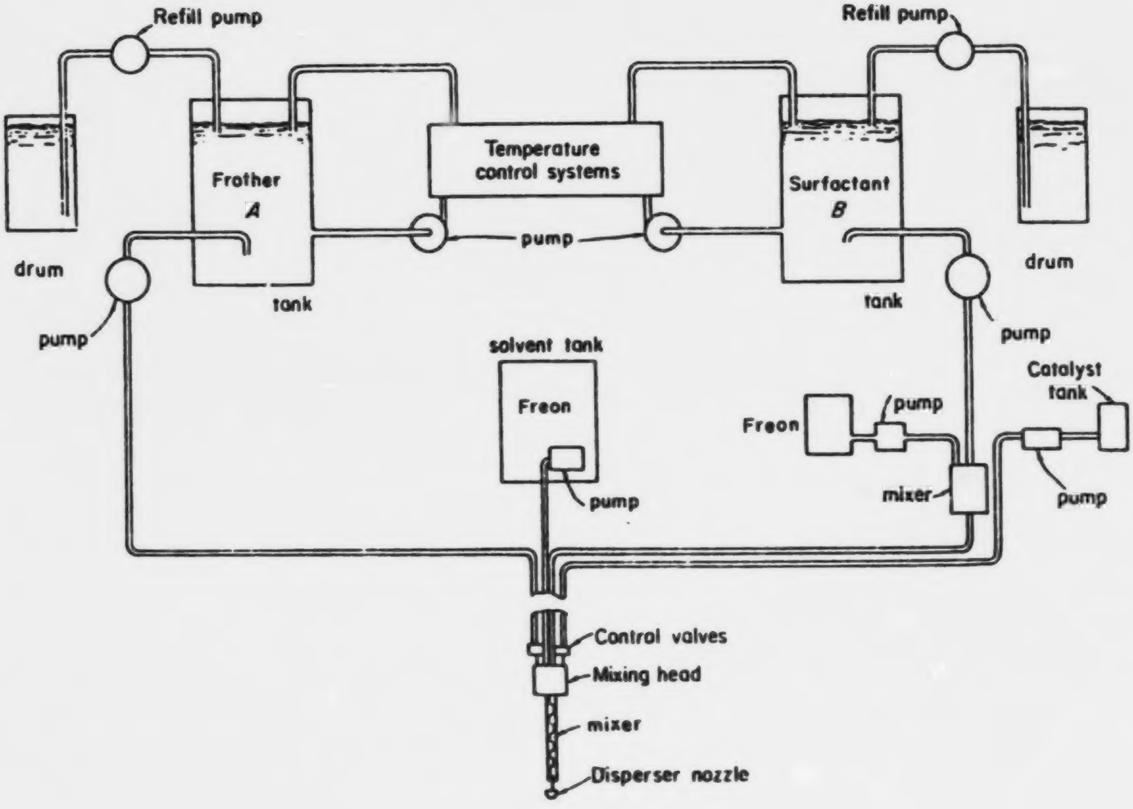
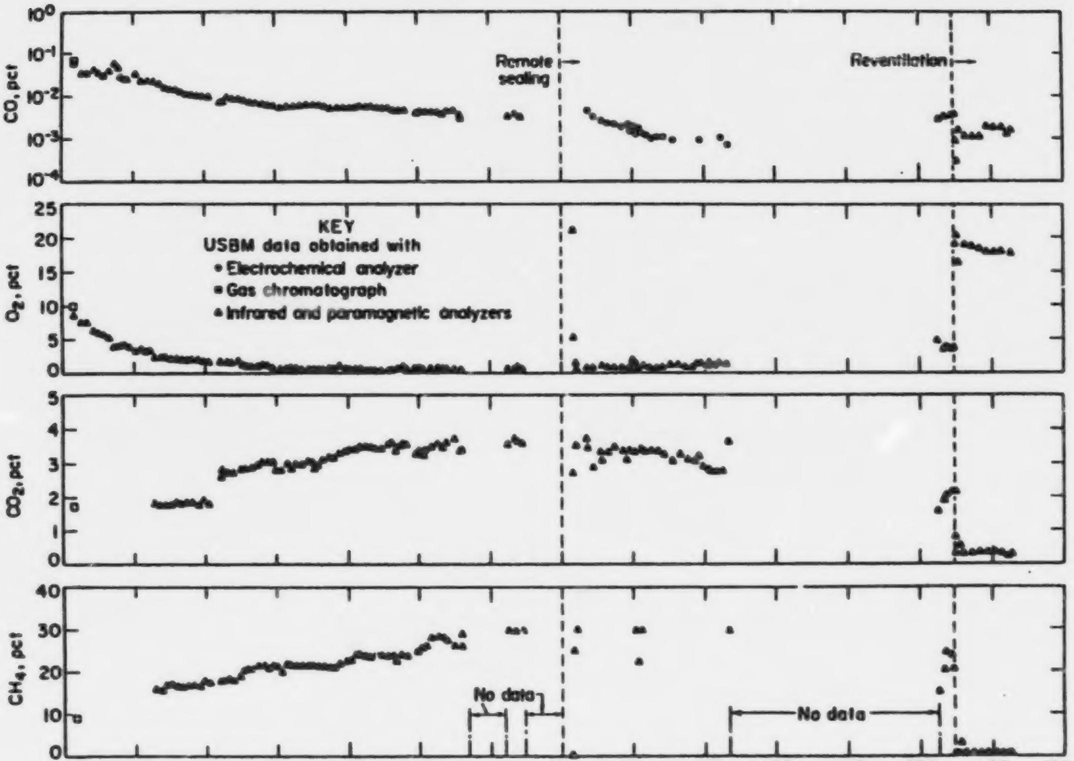


FIGURE 18-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

CARBON MONOXIDE, OXYGEN, CARBON DIOXIDE AND METHANE GAS ANALYSES VERSUS TIME FOR JOANNÉ MINE FIRE

(RD-4 borehole samples)



Case Study #19

U.S.A.

OLGA MINE

TYPE OF MINING	Room-and-Pillar
DATE OF INCIDENT	1975
TYPE OF FIRE	Coal fire in worked out area
INERTING AGENT USED	Nitrogen foam
DELIVERY SYSTEM	Surface boreholes
GAS DELIVERY RATE	32 m^3/min average over 24 hours
TOTAL QUANTITY NITROGEN USED	534 000 m^3
SPECIAL FEATURES	High quantities of inert foam used to control the fire from the surface
REFERENCE	Stone, C.B.; Hickam, H.H. and Denton G.H. 1978: **'Olga Coal Company's 5 West Mine Fire'**: Min. Cong. Journal, January 1978, p. 51-57.

1. BACKGROUND

The Olga Mine in McDowell County, West Virginia, has worked the Pocahontas No. 4 Coal Seam by room-and-pillar methods since 1914.

Mining in Olga's 5 West Section, the area of the fire, began in 1934. This was a bleeder entry for the 4 West Panel, with a barrier pillar left to protect the bleeder entries from the mined 4 West pillars. The barrier was being developed using continuous miner and shuttlecars loading into mine cars. The barrier showed signs of loading due to the heavy cover (300 m) and on February 6, 1975 a severe bump occurred. Development of the panel was stopped and retreating the panel commenced.

On February 19, 1975 it was noticed that a timber was on fire in the gob area near the miner. The cause of the fire was not determined, but may have been the result of welding in the area. Efforts to fight the fire with water were abandoned when a roof fall covered the burning timber.

The following day the CO concentration in the ventilation shaft reached 6 ppm. All men were ordered from the mine except those in 5 West Section who prepared dams and pumps in an effort to flood the fire area. By February 22, the CO content had risen to 33 ppm and the mine was closed. It was considered impossible to seal the mine because of the extent of the workings and the number of stoppings required. Fighting the fire underground was impracticable because:

- after repeated roof falls, the 40-year gob area was no longer accessible and the fire was buried
- the ventilation system could not be stopped due to the potential hazards from methane gas
- water could not be maintained at a level to flood the area due to the geometry of the mine

- the presence of methane and CO posed a severe hazard to personnel.

The decision was therefore taken to drill vertical holes from the surface and inject inert foam mixed with non-combustible solids.

2. FOAM SUPPLY AND DISTRIBUTION

Figure 19-1 shows the field equipment used for foam generation, which was based on oil and gas well fracturing technology, developed by Minerals Management Inc.

The equipment had the capacity to generate 4000 m^3/h of foam. The foam is produced by mixing nitrogen, water and foaming agent. The foam, when injected into drillholes in the fire area drives out oxygen and blankets the fire area with an inert atmosphere.

3. OPERATIONAL PROCEDURE

Five holes were drilled into the 5 West Area over the period February 26 - March 24, 1975 (Figure 19-2). #1 hole was drilled south of the fire area to set up a foam barrier between the fire and the large gas area. #2 hole was drilled to protect the eastern boundary. #3 and #4 holes were selected to provide for a possible damsite made of fly-ash or foam concrete in the event flooding was required. #5 hole was selected to directly treat the fire with foam.

Foam was injected into each borehole as soon as the drillhole had reached the seam and had been cased. The effect on CO content at the ventilation shaft is shown in Figure 19-3. By March 29, 1975 the CO content had reduced to almost zero and two days later the mine went back into production. From March 31, when foam injection was discontinued to May 2, fly-ash and rock dust were flushed down hole #5.

The quantities of materials used is summarized as follows:

Hole #	Water M^3	Nitrogen m^3	Foam m^3	Fly Ash m^3	Rock Dust m^3
1	3 065	5 380	4 540		
2	28 300	261 450	188 490		
3	730	27 170	22 700		
4	2 520	94 000	77 020		
5	7 180	145 600	122 480	16 660	800
Totals	41 795	533 600	415 230	16 660	800

One unexpected problem that arose was a foul odour in the mine, which forced its closure from April 9 - April 17. Treatment of mine water with activated carbon solved the problem.

After the mine reopened small amounts of CO were still being detected through Holes #4 and #5, so attempts to locate the source were initiated underground. A mining machine was used to mine into the area of #5 hole. Some hot coals were located and hand loaded before being cooled with water. No further evidence of fire was found and on February 24, 1976 the mine was cleared of all official orders.

The foam treatment cost $2.5 million in direct costs.

4. COMMENTS

This case study demonstrates how nitrogen incorporated into a foam and pumped down boreholes from the surface can be used to extinguish a fire in an area that cannot be effectively tackled underground.

FIGURE 19-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FIELD EQUIPMENT FOR FOAM GENERATION

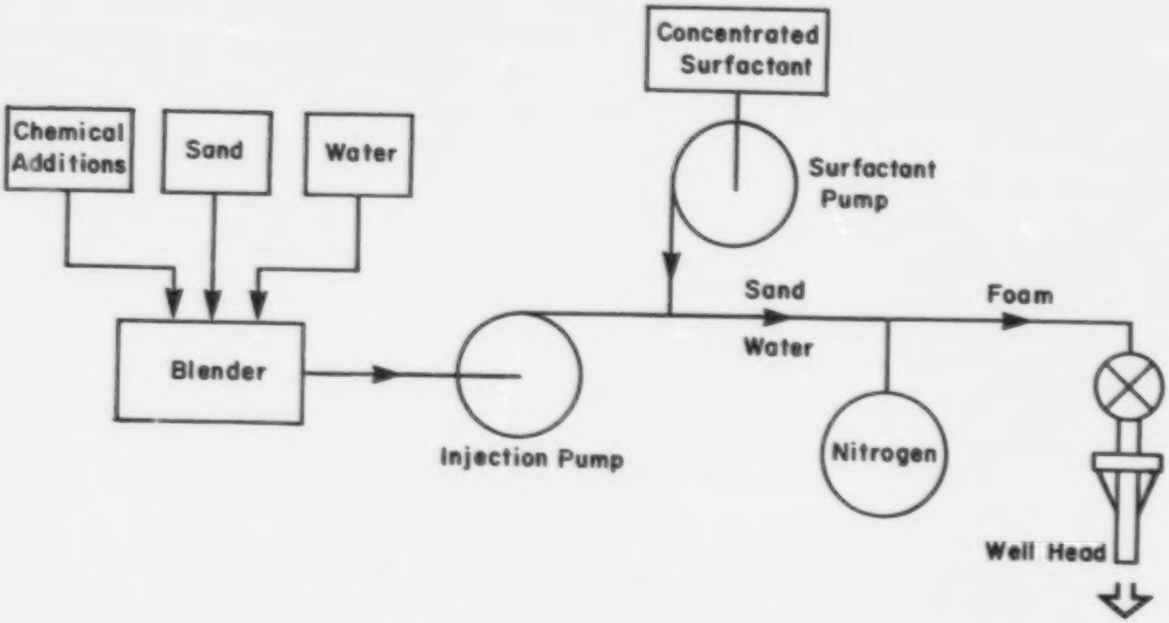


FIGURE 19-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

5 WEST SECTION FIRE AREA - OLGA COAL CO.

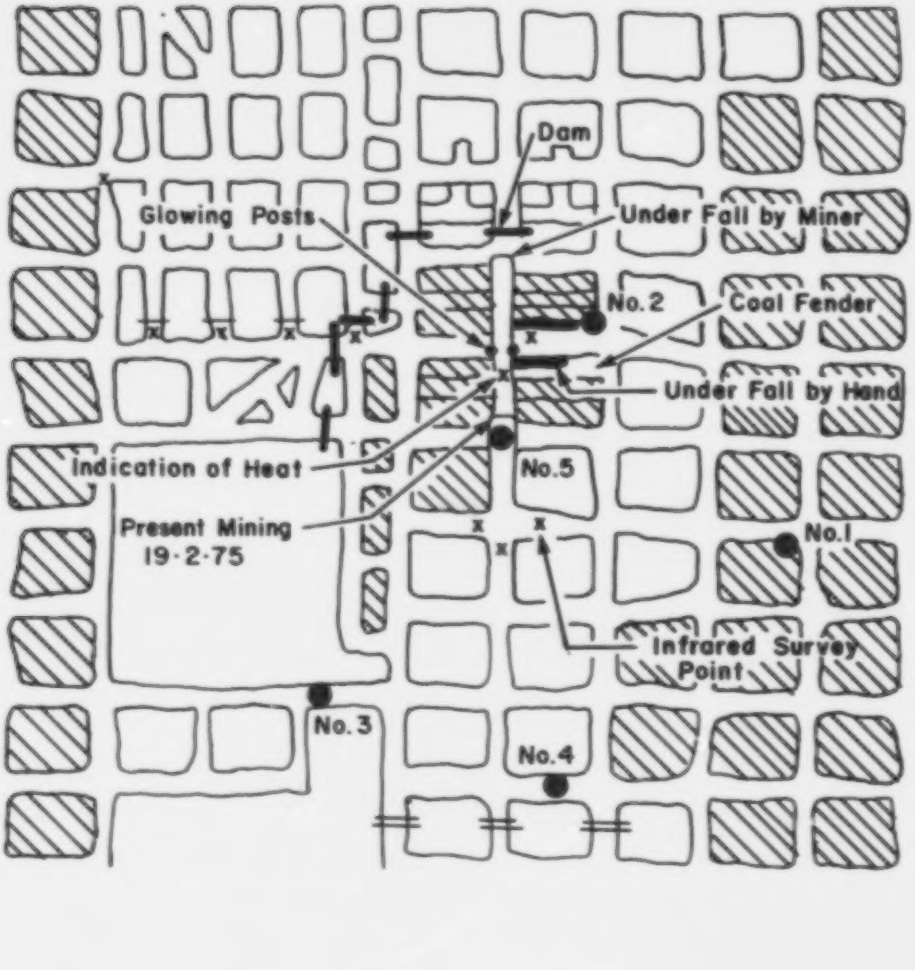
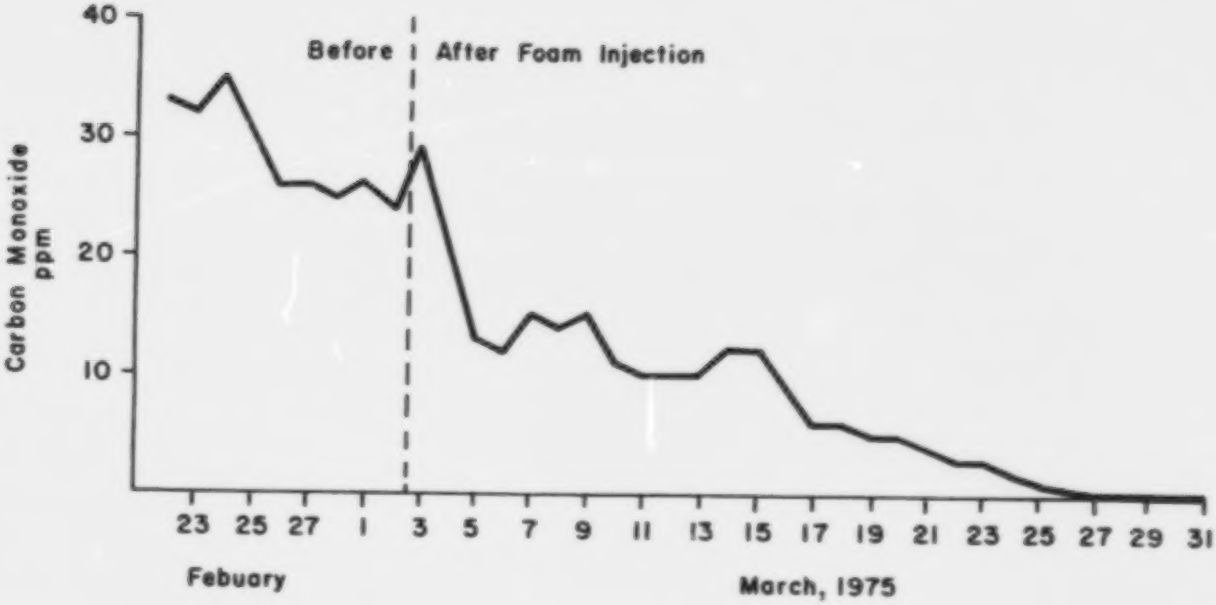


FIGURE 19-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

CARBON MONOXIDE CONTENT AT VENTILATION SHAFT



U.S.A.

PEABODY NO. 10 MINE

TYPE OF MINING	Room-and-Pillar
DATE OF INCIDENT	1981
TYPE OF FIRE	Gob fire in main entries
INERTING AGENT USED	Carbon Dioxide
DELIVERY SYSTEM	Liquid carbon dioxide injected through surface boreholes
GAS DELIVERY RATE	Variable
TOTAL QUANTITY INERT GAS USED	46 t liquid CO_2 (25 000 m^3 gaseous CO_2)
SPECIAL FEATURES	
REFERENCE	Conner, L.D. 1981: **Noninjury Underground Gob Fire;** Peabody No. 10 Mine (ID No. 1100585), U.S. Dept. of Labour, MSHA, Rept. Invest. (Aug. 17).

1. BACKGROUND

The Peabody No. 10 Mine, located in Christian County, Illinois, works the 2.1 m thick Illinois No. 6 Seam by conventional room-and-pillar methods.

A smoldering gob fire was detected under massive falls of roof in the 103-108 crosscut of the No. 1 and No. 2 main north entries (Figure 20-1) approximately 5:30 pm on August 17, 1981. All employees, except those needed for firefighting operations, were evacuated from the mine.

2. CARBON DIOXIDE SUPPLY AND DISTRIBUTION

Liquified carbon dioxide supplied in tankers was pumped through cased boreholes drilled from the surface.

3. OPERATIONAL PROCEDURE

An initial evaluation of the fire area revealed that the roof of the No. 1 and No. 2 main north entries and intersecting crosscuts had fallen to a height of 25-30 feet, and the falls extended a distance of 175 feet in the No. 1 entry and 280 feet in the No. 2 entry. Due to the extent of the falls, it could not be determined with any degree of certainty which entry or crosscut contained the fire. Instrument readings of the air coursing over the fall area revealed a CO content of 3000 ppm, 0.1% CH_4 and 14% O_2.

It was decided to load the falls from the entries and extinguish the fire by the direct attach method. Loading operations, using scoop tractors, commenced at approximately 11:00 pm and in the intervening time, temporary curtain and metal stoppings were installed around the fire area with regulators on the return side to limit the quantity of fresh air coursing over the fire area. A continuous mining machine was brought into the fire area on August 18, 1981 and used as the primary means for loading.

Because of the enormous amount of rock to be loaded and the installation of steel arches on 1.3 m centres for roof control, the loading was an extremely slow process throughout the entire operation.

Loading and roof control operations continued in the No. 2 entry until a 55 m length of the entry had been cleared of the fallen material. On August 24, it was decided to start simultaneous loading in both the No. 1 and No. 2 entries.

Hot rock and charred coal ash were encountered in the No. 1 entry approximately 6 m inbye the point where loading operations started. After loading another 3 m, a 0.8 m x 1 m glowing red hot spot was uncovered in a rider coal seam approximately 1.5 m above the seam of coal that was originally mined from the entry.

The smoldering fire was controlled with applications of water, and the hot materials were loaded from the No. 1 entry for another 4.5 m before the loading and roof control operations had to be discontinued on September 2. The roof at the face was approximately 15 m high at this point and was continuing to fall as the underlying materials were loaded out.

While the hot material was being loaded from the No. 1 entry, seven holes were drilled from the surface into the fire area inbye the rock face and 180 m^3 of a mixture of 'fire out' and water were pumped into the entries and crosscuts in an effort to cool any undiscovered hot areas.

After the hot spot was found to be in the No. 1 entry, loading operations had been discontinued in the No. 2 entry but were resumed on September 2 and continued until the morning of September 4. At that time, due to unstable roof conditions, it was decided to seal the fire area (Figure 20-2).

Nine permanent seals were constructed, three of which contained sampling pipes, around the fire area (Figure 20-3) and the sealing was completed at 10:30 pm on the same date. At this time, all workmen were removed from the mine and electrical power disconnected.

Over the next 36 hours 46 t of CO_2 were pumped into the sealed area through five of the previously cased boreholes. The first samples of the atmosphere behind the seals were collected on September 6 at 9:00 am. An analysis of the samples collected revealed the following concentrations of gas:

O_2	1.3%
CO_2	91.7%
CH_4	.9%
CO	0.5%
N_2	5.5%

Additional samples were collected three times every 24 hour period and when no significant changes were detected the mine was allowed to resume full production activities on September 8, 1981.

4. COMMENTS

This is an example of the use of carbon dioxide fed from the surface via boreholes to inert a sealed area in a room-and-pillar mine. Samples indicated that the method of combat reduced the oxygen level in the sealed area to a very low level.

FIGURE 20-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

MAIN NORTH ENTRIES AT TIME OF FIRE
Peabody No. 10 Mine

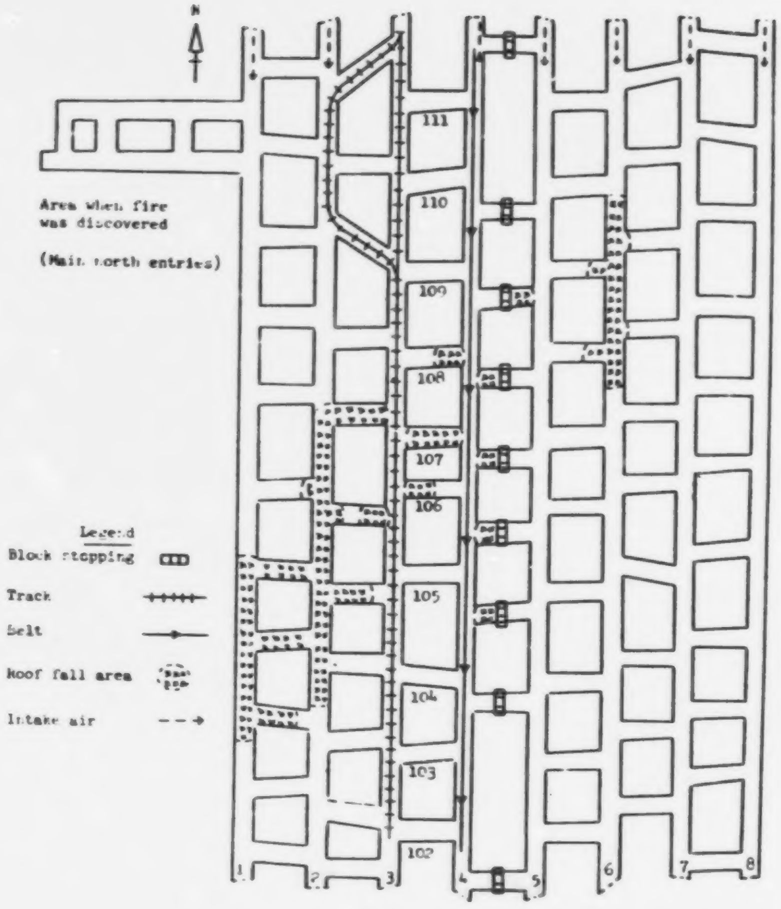


FIGURE 20-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

MAIN NORTH ENTRIES PRIOR TO SEALING
Peabody No. 10 Mine

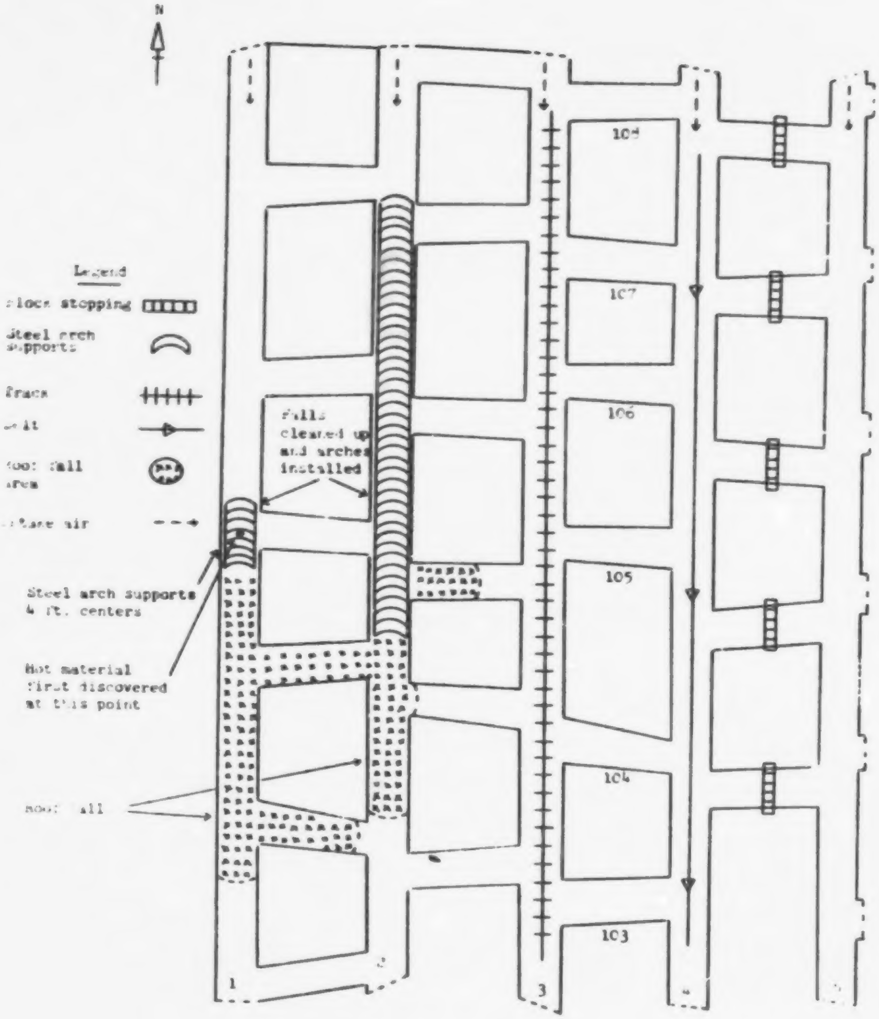
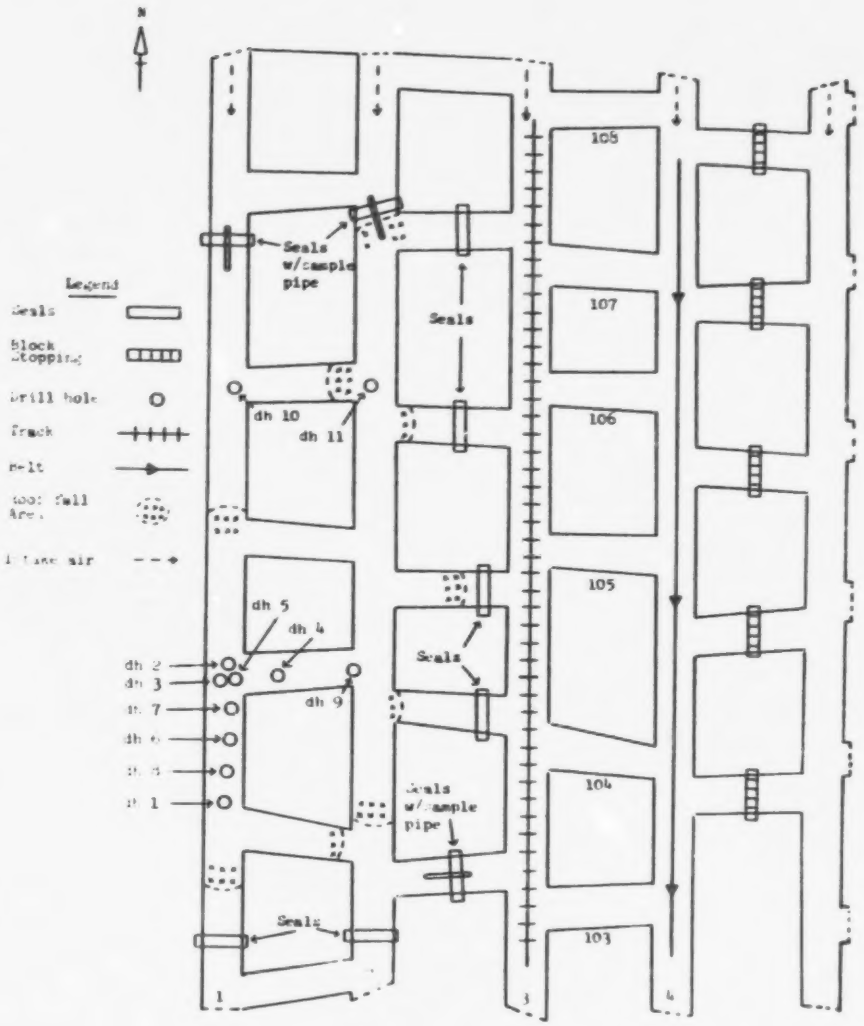

N
Legend
Block stopping
Steel arch supports
Track
Belt
Roof fall area
Intake air
Falls cleaned up and arches installed
Steel arch supports 4 ft. centers
Hot material first discovered at this point
Roof fall
108
107
106
105
104
103
1
2
3
4
5

FIGURE 20-3
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

LOCATION OF BOREHOLES AND FIRE SEALS
Peabody No. 10 Mine



USA

RIVER KING MINE #1

TYPE OF MINING	Room-and-Pillar
DATE OF INCIDENT	1979
TYPE OF FIRE	Fire in mined-out area
INERTING AGENT USED	Carbon Dioxide
DELIVERY SYSTEM	Liquid carbon dioxide injected through surface boreholes
GAS DELIVERY RATE	Variable
TOTAL QUANTITY INERT GAS USED	96 t liquid CO_2 (52 000 m^3 gaseous CO_2)
SPECIAL FEATURES	
REFERENCE	Plab D.R. 1979: Spontaneous Combustion Gob Fire; River King Underground Mine No.1 (ID No. 110725), U.S. Dept. Labour, MSAH, Rept. Invest. January 17.

BACKGROUND

The River King Underground #1 Mine, owned by the Peabody Coal Company is located in St.Clair County, Illinois. The 2.0 m thick Illinois No.6 Seam is worked by conventional room-and-pillar methods.

On Wednesday, January 17, 1979, foul odour and smoke were detected in the return air course of the 5th West panel off the sub main North (Figure 21-1). At approximately 10 pm, management notified all persons to evacuate the mine. An examination of the area by investigators did not reveal the exact location of the fire, but as nearly as could be determined the fire was located in worked out rooms where numerous roof falls had occurred. Mining in this area had been conducted during the time period of May 1, 1977 to December 1, 1977. Checks with a carbon monoxide detector as near to the fire area as possible revealed that the atmosphere contained 35 ppm of CO.

It was determined that the safest, most efficient and quickest way to control the fire would be to erect six seals, sealing off the worked out area of the 5th panel West off the sub main North (Figure 21-1) and to inject carbon dioxide into the sealed area through surface boreholes.

2. CARBON DIOXIDE SUPPLY AND DISTRIBUTION

Liquified carbon dioxide, supplied in tankers, was pumped down two boreholes drilled from the surface into the fire area.

3. OPERATIONAL PROCEDURE

The construction of six temporary concrete block seals, with joints mortared and hitched into the roof, ribs, and floor, was started at approximately 3 am on January 18, and completed at 8 pm the same day.

Two-inch plastic tubing was placed in the No.1 seal and extended inbye for a distance of approximately 20 feet. Another two-inch plastic tube was installed in the No.6 seal and extended inbye for a distance of 150 feet. The tubes were installed for the purpose of collecting samples of the atmosphere behind the seals.

During the time of construction of the temporary seals, two boreholes were being drilled from the surface for the purpose of injecting carbon dioxide into the fire area. After completion of the temporary seals, liquified carbon dioxide was pumped into the fire area beginning at 11:45 pm on January 18, and ending at 10:40 pm on January 19, 1979. A total of 80 t of liquified carbon dioxide had been used up to this point.

The mine was entered on January 20, 1979 and a check of the seals was made and samples of the atmosphere behind the seals were collected. Tests revealed that the atmosphere behind No.1 seal was 5% oxygen and 15 ppm carbon monoxide. Behind No.6 seal the atmosphere was 15% oxygen and 10 ppm carbon monoxide. An additional 16 t of liquified carbon dioxide was injected into the fire area. Operations of the mine resumed on January 22, except in the sub main North off the main East area of the mine where the construction of permanent type seals was allowed to begin.

The results of analysis collected on January 22, showed No.1 seal at 4.88% oxygen, 2 ppm carbon monoxide, and .25% methane; No.6 seal at 10.91% oxygen, 3 ppm carbon monoxide and .21% methane.

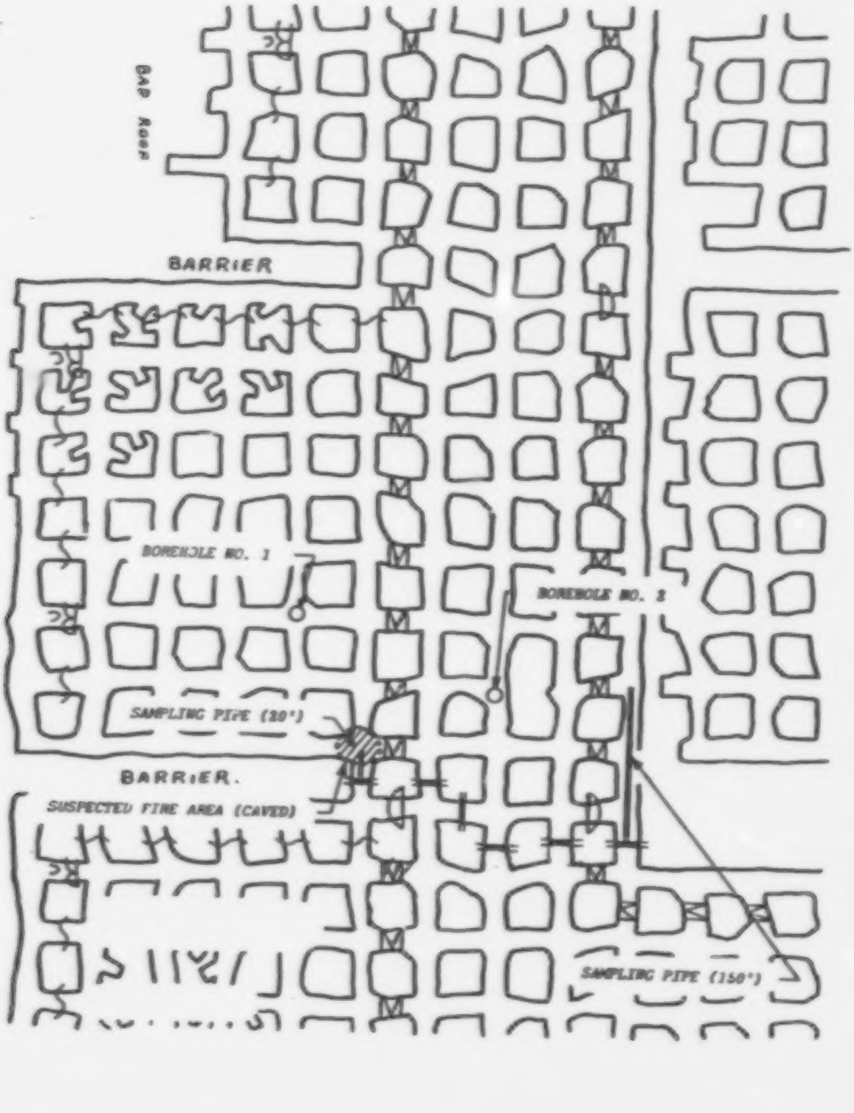
Permanent seals consisting of three rows of solid concrete blocks with all joints mortared and hitched into the ribs and floor were completed on January 25, and operations in the sub main North district resumed.

4. COMMENTS

Another example of sealing a fire area in a room-and-pillar mine and reducing the oxygen level in the sealed area by introducing liquid carbon dioxide from the surface.

FIGURE 21-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

LAYOUT OF AREA OF FIRE - RIVER KING UNDERGROUND MINE #1



Case Study #22

USSR

CONARNAYA MINE

TYPE OF MINING	Longwall
DATE OF INCIDENT	1980
TYPE OF FIRE	Heating in goaf of retreating longwall
INERTING AGENT USED	Liquid nitrogen foam
DELIVERY SYSTEM	From surface through borehole
GAS DELIVERY RATE	20 - 30 m^3/min foam
TOTAL QUANTITY INERT GAS USED	45 t liquid N_2 (24 000 m^3 gaseous N_2)
SPECIAL FEATURES	First use of liquid nitrogen foam through borehole in USSR
REFERENCE	Kozlyuk et al. 1981: **Fighting Underground Fires with Inert Foam on the Basis of Liquid Nitrogen;** (in Russian), BEZOP. RT. PROM-STI., (3), (Mar), p. 8-10.

1. BACKGROUND

A panel in the Contarnaya Mine in the USSR was worked by retreat longwall methods equipped with UKU-06 type roof supports, shearer loader and face conveyor. The face was 150 m long in a 1.6 m thick seam which dipped at 0-2°. A schematic layout of the panel is shown as Figure 22-1.

Rises in the temperature and CO concentration in the return airway indicated that a fire had started in the caved area behind the longwall face, and was increasing in intensity daily. Because of the location of the fire it was impossible to extinguish by active methods such as water hoses.

It was decided to constrain the fire area by constructing stoppings in the intake and return airways, and then to inject inert foam into the fire area through a surface borehole.

2. NITROGEN FOAM SUPPLY AND DISTRIBUTION

The nitrogen foam produced on the surface (Figure 22-2), was made up of water, foaming agent, nitrogen gas and liquid nitrogen. It was forced down a borehole drilled into the fire area.

3. OPERATIONAL PROCEDURE

Fore stoppings were erected in both the intake and return airways to constrain the fire zone. Water was sprayed at the return gate junction of the face. After the borehole was completed, foam was injected into the caved area at a rate of 20-30 m^3/min. A total of 40 000 m^3 foam were applied requiring 45 t liquid nitrogen, 20 t of foaming agent and 140 m^3 of water. This allowed the fire to be extinguished, but the temperature in the area remained high. To cool the fire zone an additional 20 000 m^3 of foam were injected. The fire stoppings were opened and ventilation re-established. One month after the onset of the fire, production recommenced.

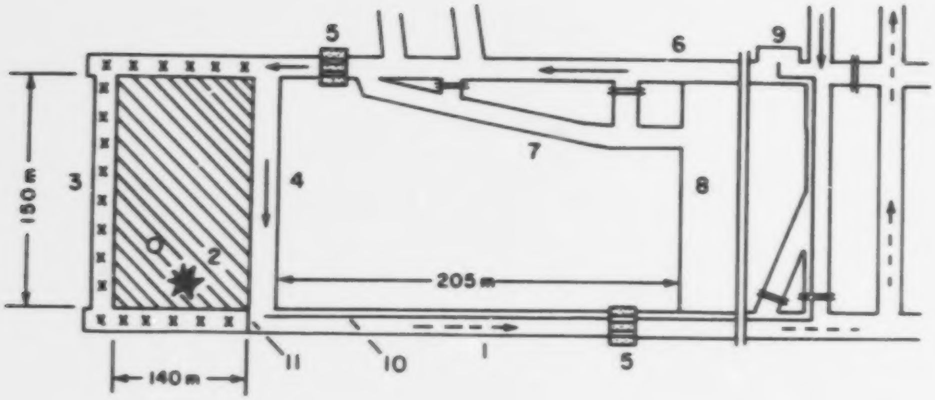
The analyses of gases at the return airway stopping, during the period of foam injection are shown in Figure 22-3

4. COMMENTS

This case study demonstrates the control of a fire in the goaf of a retreating longwall face by injection of nitrogen foam from the surface through a borehole. Access to control the fire by other methods was impossible. The liquid nitrogen was also used to cool the fire zone after the actual fire had been extinguished. It was estimated that the method used reduced the time to extinguish the fire by 5 - 10 times, compared with the conventional method of sealing off the district.

FIGURE 22-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

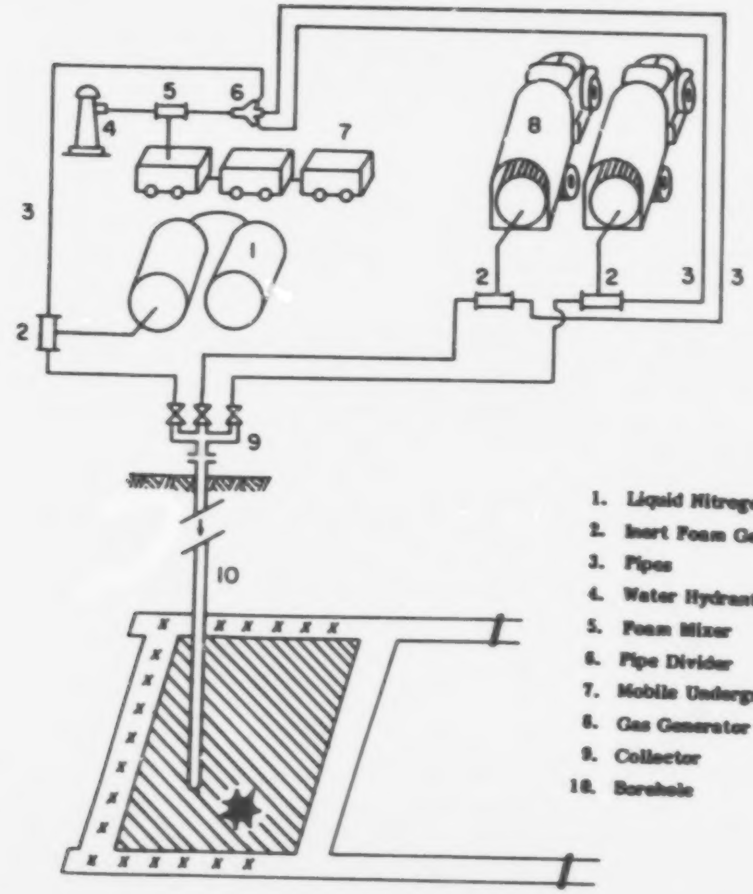
SCHEMATIC LAYOUT OF THE FIRE AREA



1. **Return Roadway**
2. **Fire Zone**
3. **Crosscut**
4. **Face Line**
5. **Gypsum Fire Stoppings**
6. **Conveyor - Roadway**
7. **Return Air Roadway**
8. **Upper Panel Roadway**
9. **Underground Extinguishing Equipment Station**
10. **Pipeline**
11. **Air Curtain**

FIGURE 22-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

FOAM GENERATION AND SUPPLY

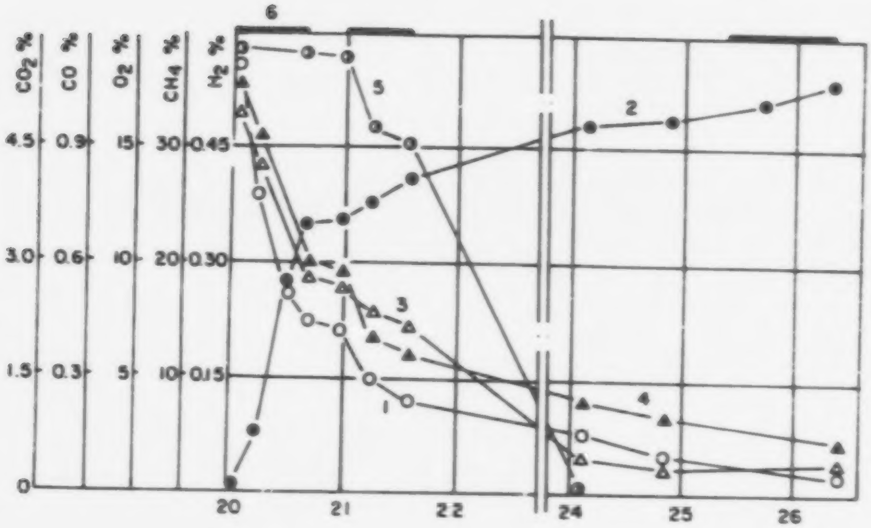


1. Liquid Nitrogen
2. Inert Foam Generator
3. Pipes
4. Water Hydrant
5. Foam Mixer
6. Pipe Divider
7. Mobile Underground Cars
8. Gas Generator
9. Collector
10. Borehole

FIGURE 22-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

ANALYSES OF MINE GASES AT RETURN STOPPING



1. O_2 - oxygen
2. CH_4 - Methane
3. CO - Carbon Monoxide
4. CO_2 - Carbon Dioxide
5. H_2 - Hydrogen
6. Foam injection

APPENDIX A

REVIEW OF INERTING EXPERIENCE TO 1974

A review of the history of inerting to 1974 appeared in the report of contract #S0231075 sponsored by the USBM and carried out by the Michigan Technological University (Appendix E). The author was Rudolf E. Greuer. An extract reproduced here with some minor editorial changes, provides a comprehensive account of inerting practice from the nineteenth century until 1974.

A. UNITED STATES

Rothwell mentions already in 1875 that steam, carbon dioxide, and mixtures of carbon dioxide with nitrogen have been injected into sealed off areas to hasten the cooling and the expulsion of the air from the enclosed space. Carbon dioxide was usually generated from mixing limestone or marble dust with sulphuric or muriatic acid or from passing air through a bed of incandescent coal, mixed with limestone. He states that the use of carbon dioxide has not been too satisfactory, but blames the defective manner of application and particularly the shortness of time allowed for the cooling of the heated fire zone for it. He feels that the district on fire should be hermetically sealed.

Spencer describes in 1898 a fire in a heading, which after sealing, kept burning slowly. He claims that the discharge of 150 lbs CO_2 into the sealed area supported the extinction of the fire greatly.

Walsh suggested in 1919 an inert gas generator based on an ordinary coal boiler furnace for fire fighting purposes. He sees the main advantage of inert gases, when applied with a positive gage pressure in sealed areas, in keeping the air out. This is especially beneficial for fighting fires in difficult to seal outcrop or other inaccessible regions.

Jones describes the extinction of a fire in the Bitner Mine, a drift mine under very light and leaky cover. The fire was caused by a grounded trolley on November 16, 1922 and had reached an extension of 1400 ft by December 16, when the application of CO_2 was commenced.

Injected into the sealed fire zone were a total of 36 tons of liquefied CO_2, equivalent to 650,000 ft^3, the major part of it between December 16 and December 25, with a maximum rate of 17,750 lb/day (160,000 ft^3/day) on December 24. From December 26 untill March 26 the CO_2 supply was continued at a reduced rate of several hundred lb/day to make up partly for the leakage losses. After January 5, when gas analyses indicated that the fire had been extinguished, the mine was partly flooded with water to accelerate the cooling process. The CO_2 concentration inside the sealed area reached a peak of 24% on December 25 and was allowed to taper off to 11%, when the fire zone was opened on March 26. The cost of injected CO_2 was estimated to be less than 10% of the total fire fighting cost. The main advantage of the CO_2 application was seen in the prevention of inward leakage of air due to pressure fluctuation and the cooling of the sealed atmosphere.

Strange reports that on February 28, 1924 a fire was discovered at the Pine Hill Coal Company near Minersville, Pa. in old workings. Sealing was completed after 4 days, but due to faulty geological conditions was not effective enough. Injection of CO_2 began March 26. Within 4-1/2 hours 194 cylinders with 50 lbs of CO_2 each were discharged, within the next 11 hours another 242, and finally within the next 10 hours an additional 122 cylinders. In May it was felt that the fire was extinguished.

Rice, Paul and von Bennewitz report that carbon dioxide had been used at the end of the last century at the Engleville Mine in Colorado to extinguish a mine fire, but without success since the fire zone couldn't be sealed well enough. The gas was generated by mixing limestone with sulphuric acid in closed vats. In another mine, the fire was extinguished by blackdamp evolving from the strata. After sealing off the whole mine with an excavated volume of 96×10^6 ft^3, the O_2 content dropped to 5% after 71 days and to 0.66% after 39 days. The authors state that they learned of other inert gas applications in mine fires also. They see the only advantage of inert gasses in keeping external gases from entering. Otherwise, the rapid absorption of O_2 in a reasonably well sealed area makes the use of inert gases, according to their opinion, unnecessary.

Forbes and Grove mention in 1948 that CO_2 is used by forcing it with perforated pipes through burning or smouldering material.

Westfield, Brumbaugh and Whittaker describe the successful use of CO_2 for fighting a coal fire of electrical origin in an intake airway. When discovered, rockfalls impeded direct fighting, so sealing was decided upon and completed 14 hours later. Only 4 seals were necessary, of which two were destroyed twice and rebuilt within the next 5 days. The sealed off area was approximately 30,000 ft^3 large. To prevent further explosions it was decided to inject carbon dioxide, which was shipped in tanktrucks to the mine, gasified at the surface and introduced through the existing waterline. A total of 5 tons or 85,700 ft^3 were introduced at an original rate of 8,000 ft^3/hr. After 1 hour the oxygen and carbon dioxide concentrations were 6.8% and 39.5%, respectively, after 10 hours, 1.2% and 73.3% respectively. Fifteen days later the sealed area was inspected and no fire being detected, ventilated. Remarkable was an increase in the CO concentration from 0.03% to 2.0% after the CO_2 injection, which had decreased to 0.23% only at the time of ventilation.

Haller and Michels report that in 1957 a fire in a timbered raise of the Penokee Iron Ore Mine near Ironwood, Michigan, was extinguished with CO_2. Immediately after the fire was detected the airflow through the raise was interrupted with temporary seals of timber and brattice cloth. Since water from the top of the raise could not extinguish the fire, the injection of bottled CO_2 into the raise from its top was begun 18 hours after the fire detection. After the introduction of some 4000 ft^3 during the next 3 hours, one ton of dry ice was dumped into the raise while the release of bottled CO_2 was continued at a reduced rate. Samples taken from behind the seals 33 hours after the detection of the fire showed that the fire had been extinguished. The CO_2 introduction was stopped after an additional 24 hours, the seals were opened 3 days later. The total CO_2 consumption was 10,000 lbs from bottles and 1 ton of dry ice. The sealed column was approximately 18,000 ft^3 - 20,000 ft^3, the length of the fire zone 40 ft.

Heers and Dennison describe the application of CO^2 to fight a fire in the Koehler Coal Mine near Raton, New Mexico. The fire was discovered on the morning of June 8, 1958. By 11:00 pm the seals in the intake and return airways to the fire zone were completed, the total sealed off volume was approximately 28,000,000 ft^3. To cool the sealed area, a closed ventilation circuit was constructed of 2 ft diameter corrugated culvert, with which gas was withdrawn from the sealed area, passed over dry ice and introduced back into the sealed area again. Liquid CO_2 was injected into the sealed area also. Over a period of 7 months, a total of 1302.5 tons of liquid and solid CO_2 were consumed. due to cracks to the surface, the O_2 content in the sealed area never fell below 3%. The fire could not be extinguished but it was possible to explore the sealed off area, drive bypass entries and to advance the seals to the edge of the fire sone. These seals were then strengthened in preparation for flooding with water, which was started on June 29, 1959. On August 11, 1959, the mine was released from the Withdrawal Order of June 1958.

Dougherty reports that during recovery operation of a caved entry section, three blocks long, indication of rekindling occurred. The entry was resealed and 'a dozen or more bottles of CO_2 were discharged into the enclosure', and the fire was extinguished. He mentions another case where 49 tons of CO_2 were pumped into a sealed area, in which rekindling during recovery operations occurred. It was, however, difficult to keep the CO_2 contained and after 10 days the seals were reopened. From his experiences with mine fires he takes the stand that the first case offered ideal conditions for the injection of CO_2 into a sealed area. 'If the area involved is of an appreciable size, the quantity and cost is prohibitive. It has been tried as an adjunct in seals on occasions, but was unsuccessful'.

McElroy concludes in 1938: 'Although a number of attempts have been made to extinguish large fires in anthracite mines by the use of steam and, in at least three instances, CO_2, the results appear to have been practically nil, or at least indeterminate'. The Miner's Circular 36 from 1948 summarizes: 'Inert gas such as CO_2 occasionally may be used to advantage in cooling or extinguishing a mine fire, however, the use of inert gases is a laborious, expensive, and often unsuccessful method. If effective seals are erected and the fire area is airtight, enough inert gases will be produced by the fire and by the consumption of oxygen to extinguish the fire within a short time. If the fire area is not sealed tightly or seals are erected, it is questionable whether the use of inert gas will be effective. Where claims of the effective use of carbon dioxide have been made, it is likely that conditions have been especially favourable, as its use is known to have been unsuccessful in several attempts'.

B. GREAT BRITAIN

The usefulness of inert gases for extinguishing mine fires was already being seriously discussed in the last century, and several applications are reported - the first one as early as 1849. Walker reports that in 1851 a fire in a coal mine at Clackmannan, Scotland, which had burned in a 9 ft seam for 30 years, was extinguished within 3 weeks with a mixture of carbon dioxide and steam. The carbon dioxide was generated in a coke furnace, cooled by water jets and then forced into the sealed fire zone with steam jets. In 1874 the Wynnstay Colliery, North Wales, was sealed after a fire broke out. Carbon dioxide was generated by mixing limestone with sulphuric acid and a total volume of

164,000 ft^3 was within 4 months forced into the downcast shaft. The atmosphere behind the seals was found to contain 5% CO_2, 25% air and 80% CH_4. Walker mentions that in 1885 one engineer claims to have used carbon dioxide on several occasions to extinguish mine fires. The carbon dioxide was generated by mixing chalk with acid in barrels and piped through the seals into the fire areas. Several months of application were usually necessary.

In 1913 carbon dioxide was used to fight a fire at the Senghenydd Colliery since due to the heavy leakage it seemed impossible to extinguish the fire by sealing it. The carbon dioxide introduction was started eight weeks after closing the seals at a rate of 400 ft^3/hr and was continued for 15 weeks. The fire could not be extinguished but sufficiently slowed down to fight it directly with water. The carbon dioxide was produced from bicarbonate of soda and sulphuric acid in a commercial generator otherwise used in the mineral water industry.

A detailed description of the use of nitrogen for fighting a coal fire in a heading, ignited by a methane explosion on July 24, 1962, has been given by Vaughan-Thomas. Due to the fact that the heading passed over a total distance of 360 ft through old workings was was exerted to large ventilation pressure fluctuations, sealing was unsuccessful. The leakage rate outbye through the seal was at first 1,700 cfm - and even after concrete spraying and building of a pressure balancing chamber, did not drop below 900 cfm. The oxygen concentration in the sealed off space of 400,000 ft^3 could not be reduced below 15.3%. On August 10 the injection of nitrogen through the seal commenced and within 60 hours a total of 1.87 million ft^3 had been introduced. The nitrogen was transported in road tankers to the mine, evaporated in steam-heated evaporators at the surface, and piped through the compressed air lines into the fire zone. Between August 15 and October 10 and November 7 the oxygen concentration could be kept at an average of 1.8% with a nitrogen input of 25,000 ft^3/hr. After a stop in the nitrogen injection between November 7 and 10, which let the O_2 and CO concentration at once rise again, nitrogen pumping was continued at 25,000 ft^3/hr until November 14, and at 40,000 ft^3/hr thereafter. Between November 16 and 18, inspection teams entered the sealed area and installed water nozzles in the fire zone. On November 25 the nitrogen flow was stopped and the heading again ventilated. When resurgence of the heating took place, a new seal, 95 yd back from the face of the heading was constructed, which was completed on November 27. Nitrogen was introduced behind this seal until December 11 at an initial rate of 40,500 ft^3/hr and at a total volume of 8 million ft^3. A reinforced concrete dam was built 5 yd outbye the second seal to flood the fire zone. Build up of an extinctive atmosphere made flooding superfluous, however. In the twenty weeks of fighting this fire, a total of 85 million ft^3 of gaseous nitrogen had been consumed. It was possible to keep the atmosphere behind the seals nonexplosive, however not possible to extinguish the fire.

Notman mentions that two and a half tons of dry ice were in another case transported into a sealed fire area to support the formation of an extinctive atmosphere.

Tideswell sees in the use of inert gases as a buffer between the air and the gas while clearing areas of potential dangers, not an unpromising approach to greater safety.

C. GERMANY

Inert gas use was more popular in the thirties than now (1974). Bredenbruch reports in 1939 that sealed fire areas, when an interest in their fast reopening exists, are flooded with carbon dioxide. This method had been used for several years. Prerequisite is a good sealing of the area and a limited extension. The carbon dioxide is always introduced at the highest point. He quotes the example of a backfilled, abandoned longwall face where spontaneous ignition caused a concealed fire. Pipes were driven into the backfill and 20,000 ft^3 of carbon dioxide sufficed to extinguish the fire. Smaller concealed heatings were frequently fought with perforated injection lances. In one case an initial quantity of 7,000 ft^3 and continuing injections of 2,300 ft^3 - 3,700 ft^3/week of carbon dioxide over a period of 8 weeks extinguished the fire.

Both reports that in the only case of inert gas application during the last decades - a rate of 35,000 ft^3/hr over several weeks - was not capable of extinguishing the fire, but allowed to advance the seals and to reduce the sealed area. The inert gas was generated in a mobile plant from heating oil, and had a composition of 15% CO_2, 80% N_2 and 5% H_2O.

D. MAINLAND CHINA

Wang Tie-Sin describes two coal mine fires, which could be extinguished with inert gases. The first fire (Colliery No. 1 Tchegan Division) started in June 1959 and required the sealing of a whole mine level with a volume of 3.5 million ft^3. Injection of a mixture of carbon dioxide and nitrogen started in March 1960, and by July 3.5 million ft^3 had been forced into the sealed area. Gas samples taken in June behind two seals showed carbon dioxide of 5.1% and 6.7%, methane concentrations of 1.0%. The carbon monoxide concentration in the fire area was in July, 0.03%. After inspecting the sealed area in September 1960, the seals were opened.

The second fire (Dalu Colliery No. 1 Tchegan Division) started in August 1960 and was immediately sealed. Injection of a mixture of carbon dioxide and nitrogen started in September 1960. After 3.5 million ft^3 had been introduced within the next 3 months into the sealed area with a volume of 1.6 million ft^3, the carbon monoxide content in the sealed atmosphere disappeared. Gas samples taken at three different seals showed carbon dioxide concentrations of 6.9%, 9.1% and 10.0%; oxygen concentrations of 2.7%, 1.0% and 0.9%.

The inert gas was in both fires generated with an intermittently operating coal furnace, charged with 1,760 lbs of coal which were consumed in 12 - 14 hours. The combustion gases contained: CO_2 16%, CO 0%, O_2 3%. It is claimed by Ratuskov and colleagues that this inert gas generator is based on work done by the Russian Research Institute for Workprotection in the Mines of the Eastern District (Vost NII).

E. POLAND

The Polish Coal Mines use inert gases for two different purposes: to support the build-up of a non-explosive atmosphere in sealed areas, and to prevent air leakage through the seals.

The inert gas requirements for extinguishing fires by lowering the oxygen concentration to approximately 3% are considered to be too large for practical applications.

Air leakage through seals is prevented by so-called insulation locks - consisting of a group of 2 - 4 stoppings, erected adjoining to the seals, and forming an inert gas-filled chamber of 1500 ft^3 - 3,000 ft^3 volume. The inert gas is kept at an overpressure of 0.4 - 0.8 in. wg. The stoppings have well fitting doors to allow fire fighting teams to penetrate into the fire zone.

The inert gas source is a truck mounted West German built generator of the type Mahler-Exo-I 200/a, with a capacity of 120 ft^3/min. The inert gas composition is 85% N_2, 13% CO_2, 2% O_2 maximum.

F. CZECHOSLOVAKIA

A thorough review of inert gas methods is given in a paper by Hajek. It contains a list of all inert gas application in Czechoslovakia up until 1969.

The first large scale use took place in 1949 at the Doubrava Colliery, where a series of methane explosions had caused widespread fires. When sealing was not able to extinguish these fires, an improvised nitrogen plant of the Claude system with a capacity of 25,000 ft^3/hr was set up at the mine. A total volume of 178.6 million ft^3 had to be released into the mine before it could be reopened.

Three more fires were fought in 1951, 1955 and 1956 with bottled CO_2, mine-site produced and trucked N_2. In 1957 three truck trailers carrying bottled nitrogen at 5,100 psi and with a capacity of 21,000 ft^3 each, were added so that now delivery rates of several 100,000 ft^3/day can be maintained in the Ostrava-Karvina Coal Basin. Where no discharge of the gas through pipelines is possible, bottles of 10 gal. volume are used for the transport between trailers and fire zone. They have a capacity of 210 ft^3 each at 2,200 psi.

Experiences with CO_2 have not been favourable. When blown into the fire, the CO content rose to 5% - 6%. Nitrogen can furthermore be obtained in larger quantities.

To limit the gas consumption only smaller sealed spaces are flooded with nitrogen to extinguish the fire. In the majority of fires, low-volume nitrogen pressure chambers are employed. These are two nitrogen filled chambers, 10 - 17 ft long and fitted with well sealing doors, which are built after the waiting period has passed, outside each seal of the fire zone. The nitrogen is kept at a pressure approximately

0.4 in. wg. higher than that of the sealed atmosphere or mine atmosphere - whichever is higher. This requires a nitrogen supply of 35 ft^3 - 70 ft^3/min per seal.

Pressure chambers have two purposes: they shall prevent a leakage of other gases than nitrogen through the seals, and they shall permit an opening of the seals to allow fire fighting teams to advance towards the seat of the fire under the cover of a nitrogen curtain. If the whole sealed area cannot be inerted, new seals are built at the farthest point of advance.

Wald brings an example for the rapid extinction of a mine fire with the aid of pressure chambers. It is estimated that the reduced air leakage through the seals allows to shorten the sealing periods from 4 - 6 months to 1 - 2 months.

In 1967 an inert gas generator, based on the thermal dissociation of ammonia and subsequent combustion of hydrogen with a capacity of 650 ft^3/min was developed by the Mine Rescue Center. It is, however, contemplated to use in the future the generator GIG-4, developed in the Soviet Union.

G. USSR

Omipov and colleagues mention that inert gases for fighting mine fires have been used since the beginning of the century. They report that in the years 1963 - 1965 nine large mine fires in the gassy mines of the Donetsk district alone were sealed with simultaneous injections of carbon dioxide to prevent explosions, and that the total amount of CO_2 consumed was higher than 3.5 million ft^3.

The final detailed description of the extinction of a mine fire which could be found in the literature is from 1960 and deals with the injection of bottled carbon dioxide into a sealed area of 210,000 ft^3. Within 2 days a total of 195,000 ft^3 were released, but due to considerable leakage, the gas composition in the sealed area changed only to 13.6% CO_2, 3.1% CH_4, 12.0% O_2 and 0.03% CO. Much more carbon dioxide had to be used before the fire could be extinguished.

Ratuskov and colleagues report that inert gas methods found - till 1964 - no wide application due to the limited gas supply. The Research Institute for Workprotection in Mines of the Eastern Districts (Vost NII) therefore developed a mobile inert gas generator with a capacity of 17,700 ft^3/hr, which consumes approximately 90 lbs/hr of heating oil and produces a gas with 1.5% - 2.5% O_2, 12.5% - 14.0% CO_2 and only traces of CO. This generator was successfully used to extinguish a fire in the gob of the Mine No.3/4 'Ziminka' (Kombinat Kuzbassugol). After sealing a mine area of 106,000 ft^3 a total of 1.1 million ft^3 inert gas was injected over a period of 3 months in rates of 17,700 - 21,200 ft^3/hr intermittently. One month after stopping the inert gas introduction, the seals were opened. The gas inside contained 5% O_2 and 4% CO_2, CO was missing. The authors see the special advantage of the inert gas application in the fact that a nonexplosive atmosphere behind the seals is established quickly so that interference of the fire with the rest of the mine is minimal.

Sevostjanov describes a mine fire where inert gas application did not have the desired result. In the Mine No. 12 'Osnovnaja' (Kombinat Donbassantracit) a methane blower was ignited by shotfiring in a longwall face and produced a flame of 100 ft length and 28 - 33 ft width. Soon the timber lagging of the return airway burned over a length of 100 ft. With ample use of water the fire could be reduced to the gas flame alone. The latter was fought with water, carbon dioxide, and chemicals from hand extinguishers - but without success. To avoid the formation of explosive mixtures, it was decided to refrain from sealing or reducing the airflow, but rather to employ the inert gas generator GIG-2. This generator produces a mixture of steam and inert gas at a rate of 635,000 ft^3/hr. Although the O_2 content could be reduced to 15% and the CO_2 content increased to 5% (after 1 hour), the operation of the generator had the effect of increasing the airflow through the fire zone and the temperature of the combustion products, so that the fire started to spread again. After several hours the use of the inert gas generator was, therefore, suspended and the fire fought again with water. The fire was finally extinguished by draining the CH_4 into gas boreholes and cooling the rock with water brought in through other boreholes.

Osipov and Orlov report that a fire which had been started by a methane ignition could in the accessible parts of a longwall coal face be extinguished by conventional means but continued to burn in the waste area. After sealing, nitrogen from liquid gas containers at the surface and four AGU-2m evaporators with a combined capacity of 1,400 ft^3/min was piped into the gob through two 4 in pipelines at a rate of 1,130 ft^3/min over the first 64 hours and at a rate of 370 ft^3/min over the next 99 hours. The total volume of nitrogen injected was 6.3 million ft^3 or 1.9 times the volume of the sealed off space. After 40 - 50 hours, the oxygen concentration had dropped to 1%. Four months after the nitrogen injection, the carbon monoxide had disappeared. After allowing an additional 2 months for cooling, the sealed area was inspected and reopened. The cooling times were calculated from heat transfer considerations, which gave an answer of 6 - 8 months.

Belik and colleagues describe the successful use of carbon dioxide to extinguish a spontaneous coal fire in the gob area of a steep longwall face. After sealing, the carbon dioxide was released from the top entry in a quantity of 164,000 ft^3 over 12 hours and at a rate of 10 ft^3/sec. The composition of the atmosphere along the face and the entries was recorded over a period of 100 hours. Although the seam had a dip of

60°, the carbon dioxide filled all mine workings and diluted the oxygen to less than 3%.

Sobolev brings in his book Mine Rescue, an example for the use of nitrogen. At the Mine No. 23 of the Karaganda Coal District, a fire in a longwall face, which gradually spread into the return airways, was discovered April 19, 1969. After 4 hours of attempts to extinguish the fire with water and foam, it had to be decided to seal the fire zone. Explosion proof seals were completed in April 23 in two intake and three return airways. Gas samples showed: 7.5% CO_2, 8.4% CH_4, 6.9% O_2, 1.4% CO, 1.6%H_2 behind return seals and 1.0% CO_2, 4.2% CH_4, 16.8% O_2, 0.7% CO, 0.3% H_2 behind intake seals.

After 25 hours the composition had changed to: 6.5% CO_2, 21% CH_4, 2.6% O_2, 1.2% CO, 1.1% H_2 behind return seals and 1.5% CO_2, 18% CH_4, 5.5% O_2, 1.0% CO, 1.2% H_2 behind intake seals, and miners were allowed to re-enter the mine.

To stop active burning and to create a stable non-explosive atmosphere, the injection of inert gas was decided upon. Used was liquid nitrogen from railroad tank-cars, which was evaporated in gasification plants of the type AGU-2m at the surface and piped through gas drainage and water lines of 6 in and 5 in diameter, and a total length of approximately 1.5 miles. Four plants AGU-2m could under these conditions feed an average of 1,240 ft^3/min into the fire zone. Between May 3 and May 7, 1969, a total of 6,000,000 ft^3 N_2 were released, or 1.24 times the sealed volume of 4,850,000 ft^3. An additional 940 ft^3 of gas were poured through a borehole into the fire zone. This changed the composition of the sealed atmosphere from 3% - 4% CO_2, 36% - 49% CH_4, 6.5% O_2, 0.07% - 0.2% CO before the N_2 release to 0.5% CO_2, 41% CH_4, 0.9% - 1.9% O_2 and 0.0% CO after the release of the nitrogen.

An unpublished report describes 7 applications of the inert gas generator GIG-2 through the years 1969 - 1972 in coal mines. In all cases the ventilation of the fire zones was maintained to such an extent that no explosive mixtures could form and was cut off only after the fire zone had been filled with an unexplosive atmosphere.

The transport of the inert gases into the fire zone has been studies by several researchers. Kessariiskii investigates the usefulness of existing pipeline systems in mines for piping nitrogen from the surface to the prospective fire sites and comes to the conclusion that the dimensions of the existing systems (water, compressed air, gas drainage) suffice. Osipov and colleagues describe an experiment to study the carbon dioxide flow within a sealed longwall face of 360 ft length, 3 ft height and 23° dip. The pertinent haulage and ventilation roadways had cross sectional areas of 54 ft^3, the ventilation was ascensional. Both roadways were sealed in a distance of 1,150 ft from the face, the airflow was regulated at 105 ft^3/sec or 2 ft/sec. Over a period of 90 minutes, carbon dioxide was released through the lower seal at a rate of 35 ft^3/sec. This caused the airflow to drop to 18 ft^3/sec after 20 minutes and to 3 - 5 ft^3/sec when the ventilation pressure, acting at the face, was further reduced. Airflow rates, methane, carbon dioxide and oxygen concentrations were monitored along the roadways and the face. The results showed the following peak values of gas concentrations:

200 ft from lower seal:	62% CO_2, 8% O_2
1,100 ft from lower seal:	45% CO_2, 11% O_2
100 ft inside longwall face:	30% CO_2, 14% O_2, 5% CH_4
Top of longwall face:	12% CO_2, 16% O_2, 14% CH_4

This indicates that the ventilating pressure acting at the longwall face was too small to move the carbon dioxide up the face. To guarantee mixing of the CO_2 with air, the authors recommended that Reynold's numbers, larger than 50,000 should be maintained.

A similar experiment was conducted with nitrogen by Gorb and Stalerov. A longwall face of 640 ft length, 3 ft height, 10 - 11° dip, which had advanced approximately 1,000 ft from a raise and been worked with full caving, was sealed in its pertinent haulage and ventilation roadway close to the raise. The total sealed space was estimated to be approximately 920,000 ft^3. Nitrogen was released through the lower seal over a time period of 260 minutes and a rate of 2,000 ft^3/min, which amounts to a total nitrogen quantity of 0.57 times the sealed volume. The airflow had been reduced by the sealing from 19,500 ft^3/min to 5,600 ft^4/min and fell to 2,300 ft^3/min after the nitrogen release. Monitored were nitrogen, oxygen and methane concentrations in 90, 880, 1400, 2150 and 2460 ft from the point of nitrogen release. The results showed that the nitrogen moved with the ventilating air and filled the sealed space, which required approximately 3 hours. Due to the low air velocity, considerable methane layering was, however, observed along the upper entry.

The losses, which are caused by leakage through the gob area when inert gas is released into the roadways of a longwall face, have been estimated by Abramov and colleagues. The results of the calculations show that the main factors are the backfill method, the height of the seam, and the length of the entries. The length of the longwall face is less significant. As is to be expected, the losses can be considerable.

Ozipov and colleagues measured the changes of carbon dioxide concentrations in sealed longwall faces and pertinent roadways after the gas introduction had been stopped. They found that up to 75% of the CO_2 disappeared within 3.5 to 14 hours and explain this fact mainly by absorption by the coal.

Grekov and Gorb bring, without an explanation of their origin, two graphs for the concentration change of CO_2 and N_2 along airways for different flow velocities. Since these changes are much larger for CO_2 than for N_2 they might be based on absorption losses. Under the assumption that a CO_2 flow rate of 1060 ft^3/min (120 cylinders per hour) is the practical maximum, that the oxygen concentration should be lowered to at least 13.5% to establish a non-explosive atmosphere. Due to air leakage, only a 60% - 70% CO_2 concentration can be expected at the intake seals. They conclude that CO_2 should be used only in sealed spaces below 140,000 ft^3 volume or 1,300 ft distance between intake and return seals. For N_2, with lower concentration changes and much larger possible flow rates, there is no limitation except for dead end workings.

The limitations in the application of carbon dioxide in non-horizontal mine workings have been investigated by Belik. To reduce the large gas quantities needed for inerting the atmosphere in sealed off fire areas, the admixture of Freon 12 and Freon 114 V2 to nitrogen and carbon dioxide has been tried in laboratory tests.

Additional experiences with the use of inert gases are contained in the Handbook for Mine Rescue, Vol. 3 and 4, guidlines for the application of inert gases in the book Mine Rescue by Sobolev and the Manual on the Isolation of Fires in Mines with Gas Hazards.

Sobolev, head of the Department Mine Rescue in the USSR Ministry of Coal Production, sees as the main advantage of the inert gas method that it allows to prevent the formation of explosive mixtures while the fire is sealed. If a direct attack is possible, water with its large heat binding property is a better means of fire extinction. The fast filling of a sealed area with an inert atmosphere allows to reduce the sealing time by approximately 50%. Three inert gas sources are used: CO_2 from cyliners - maximum delivery rate 1250 - 1400 ft^3/min; liquid N_2

from tankcars - transported and evaporated at the mine site in evaporator trucks - like the type AGU-2m with a transport capacity of 50 ft liquid nitrogen and a delivery rate of 2800 - 3500 ft^3/min gaseous nitrogen, which is piped from the surface into the fire zone; (another evaporator frequently used is the type PeEAS-100); and, inert gas from portable generators. The latest development is the type GIG-4 of which presently 4 are used mainly in the Donjetsk District. It has a capacity of 12,000 ft^3/min, weighs 2000 lbs and can be dissembled and assembled for underground transport and used in a very short time. The composition of the generated inert gas is: 1% O_2, 8% CO_2, 500 ppm CO, 51% N_2, rest H_2O, the discharge temperature is 175 - 195°F, the gauge pressure 40 in wg. Series production of this generator is under discussion. A smaller generator with 1750 ft^3/min capacity for replacing inert gas losses in sealed areas is under construction.

Right now liquid nitrogen is still the inert gas source mainly used in the USSR. If the inert gas is applied early enough, the seals of open fires can be opened after 2 - 3 weerks. For concealed fires, up to 6 months may be necessary. No forced recirculation of inert gases is used.

A foreign expert estimates the number of inert gas generator uses in the USSR to be approximately equal to 60.

An unpublished report of a visit to the Donetsk District in 1971 states that every mine rescue centre is equipped with 4 trucks loaded with liquid nitrogen and 1 evaporator. The nitrogen is mainly used to inert explosive gases. The capacity of each truck is 7.5 tons, the rate of discharge 350 ft^3/min. Additional nitrogen is usually transported by rail to the mine. It is planned to acquire 15 ton trucks for mines without railroad connections.

APPENDIX B

MINE FIRES

B.1 GRAPHICAL METHOD OF DETERMINING EXPLOSIBILITY DATA

When the computer is not available the Hughes and Raybold graphical method should be used to calculate the explosibility data. The necessary graphs are included, Graphs (Figures B-1, B-2, B-3 and B-4), together with a report form (Figure B-5) proceed as follows:

1. From the analysis calculate (a) the proportion of methane and (b) the proportion of hydrogen in the combustibles.

2. Place a transparent cross cursor over Graph 1 (Figure B-1), lining up the percentages of methane and hydrogen in the combustible gases with the scales on the outside of the square. Where the intersection of the two axis of the cross falls on the scale printed inside the square is the combustible gas content of the lower explosive limit mixture with air.

3. In similar manner find the oxygen content of the nose limit from Graph 2 (Figure B-2), and the combustible gas content of the nose limit from Graph 3 (Figure B-3). The combustible gas content of the upper explosive limit mixture with air is read off the ordinate of Graph 4 (Figure B-4) in the usual way.

Mark all these points on the standard Graph (Figure B-5) and construct the explosive triangle.

Example:

Actual analysis obtained	**%**
Carbon dioxide	5.32
Oxygen	11.89
Nitrogen	75.46
Methane	1.93
Carbon monoxide	1.40
Hydrogen	4.00
The combustibles - methane, carbon monoxide and hydrogen - total	7.33
Proportion of methane in combustibles	
$= \frac{1.93}{7.33} \times 100$	26.3
Proportion of hydrogen in combustibles	
$= \frac{4.00}{7.33} \times 100$	54.6
From Graph 1 (Figure B-1), lower explosive limit in air, combustible content	5.0
From Graph 2, (Figure B-2), nose limit oxygen content	6.9
From Graph 3, (Figure B-3), nose limit combustible content	5.4
From Graph 4, (Figure B-4), upper limit in air, combustible	34.6

The Hughes and Raybold method gives a most important facility for determining the nature of the fire gases and is an essential safeguard to the safety of personnel fighting the fire, and gives a strong indication of the safest and most efficient way of combatting it.

FIGURE B-1
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

GRAPH 1: COMBUSTIBLE CONTENT AT THE LOWER EXPLOSIVE LIMIT OF MIXTURES OF METHANE, HYDROGEN AND CARBON MONOXIDE IN AIR

Carbon Monoxide % = 100 – (%CH_4 + % H_2)

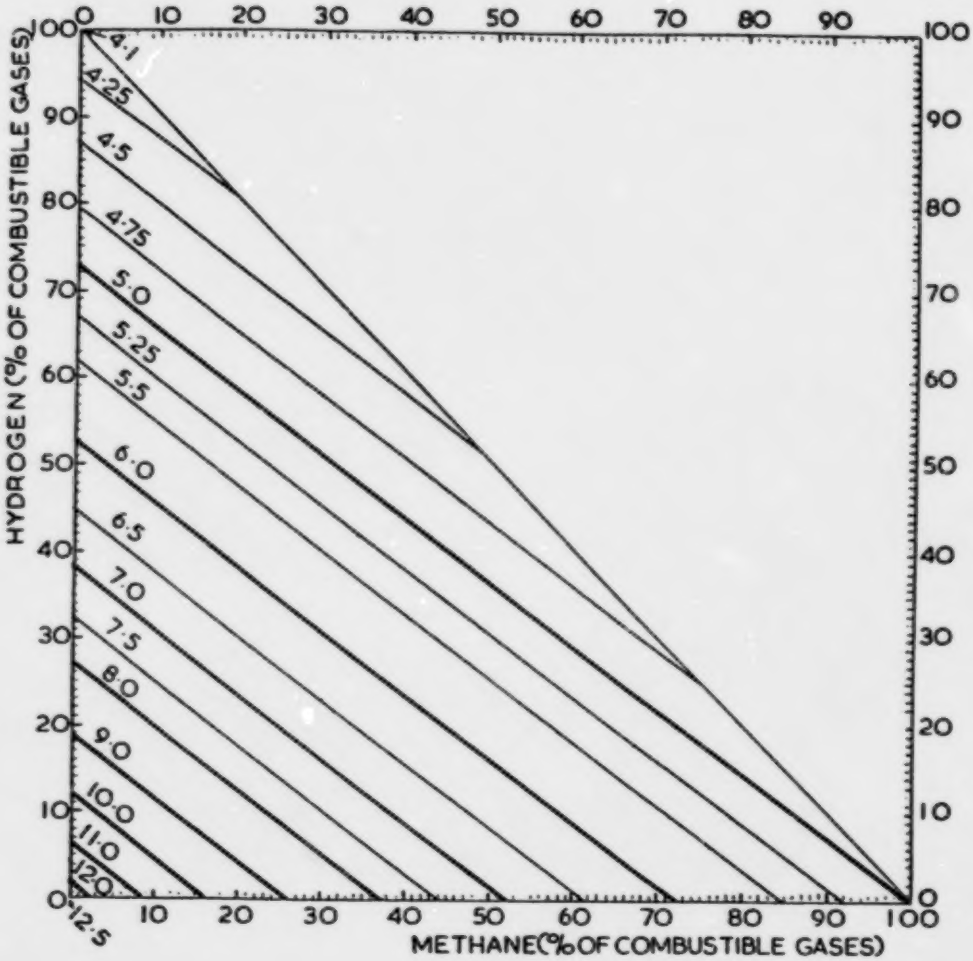


FIGURE B-2
UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

GRAPH 2: OXYGEN CONTENT AT THE NOSE LIMIT OF MIXTURES OF METHANE, HYDROGEN AND CARBON MONOXIDE IN AIR AND EXCESS NITROGEN

Carbon Monoxide % = 100 − (%CH_4 + % H_2)

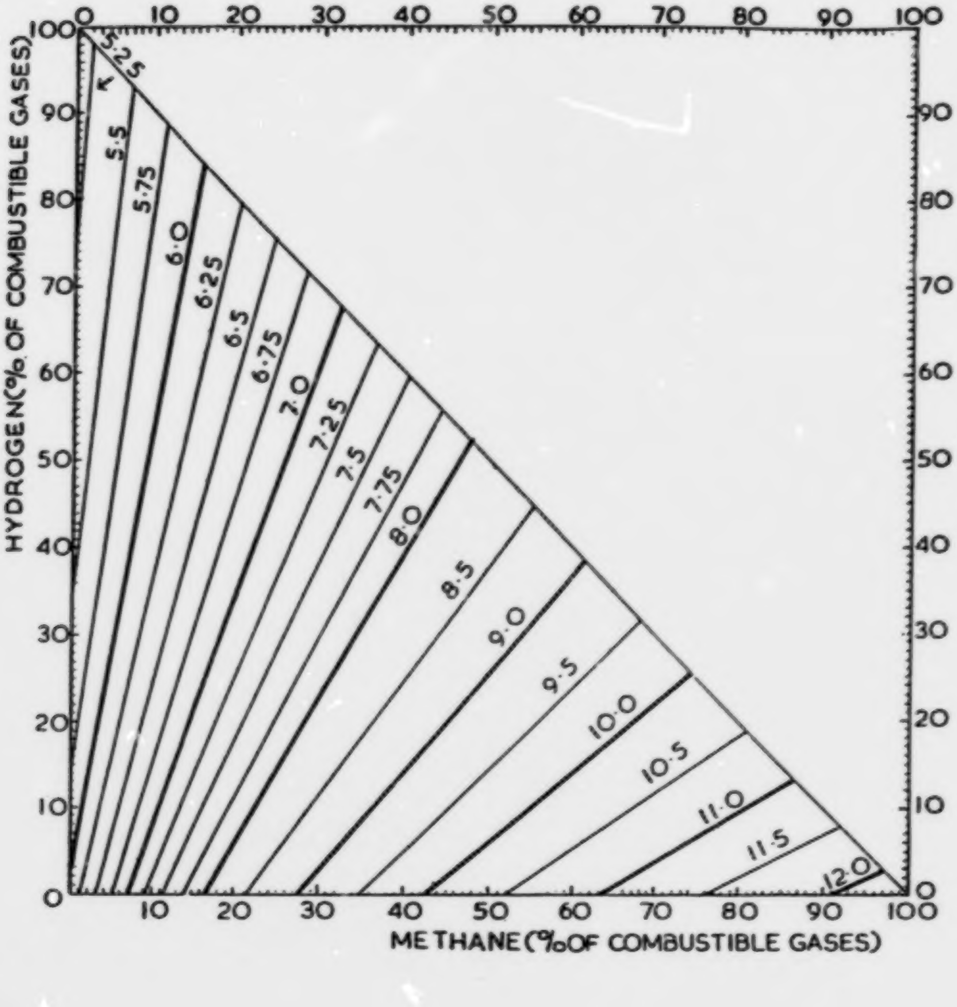


FIGURE B-3

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

GRAPH 3: COMBUSTIBLE CONTENT AT THE NOSE LIMIT OF MIXTURES OF METHANE, HYDROGEN AND CARBON MONOXIDE IN AIR AND EXCESS NITROGEN

Carbon Monoxide % = 100 - (%CH_4 + % H_2)

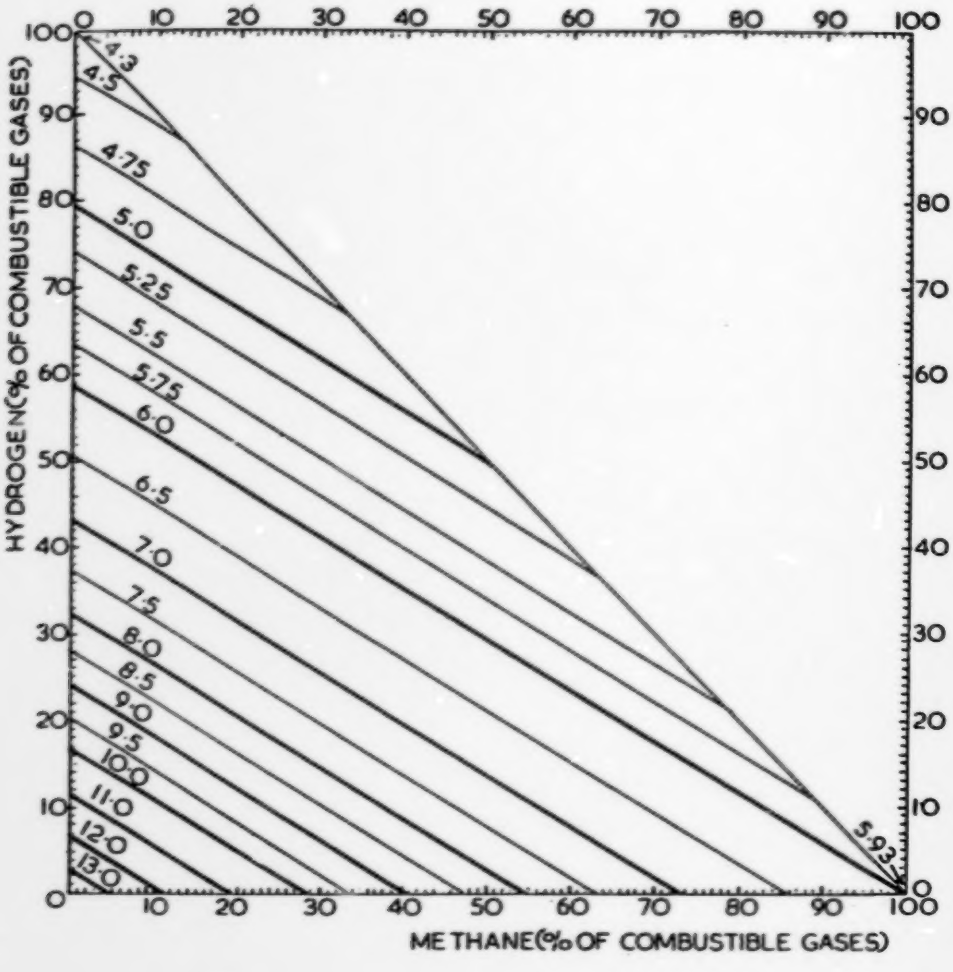


FIGURE B-4

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

GRAPH 4: COMBUSTIBLE CONTENT AT THE UPPER EXPLOSIVE LIMIT OF MIXTURES OF METHANE, HYDROGEN AND CARBON MONOXIDE IN AIR

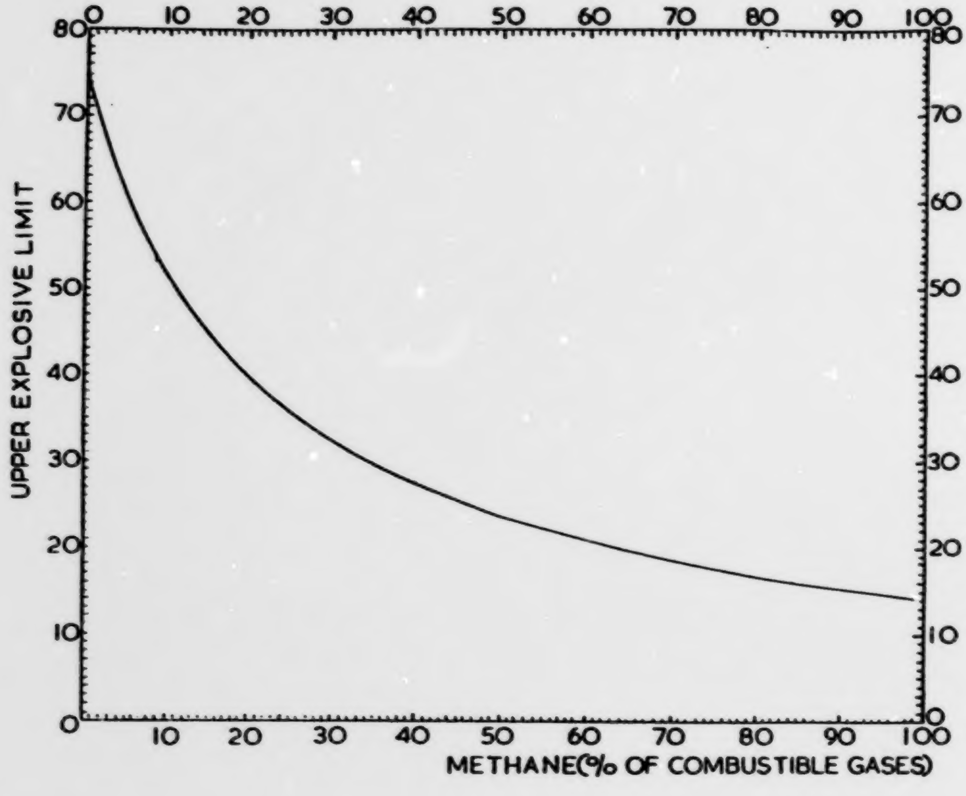


FIGURE B-5

UNDERGROUND COAL MINE FIRE CONTROL WITH INERTING SYSTEMS

REPORT SHEET

EXPLOSIBILITY CALCULATION (Raybold & Hughes Method)

1. Colliery:- | 2. Date:- | 3. Time:-

4. Sample Location:-

5. Composition of Sample:- % By Volume

Oxygen (A) = ________

Methane (B) = ________

Hydrogen (C) = ________

Carbon Monoxide (D) = ________

6. Calculation:-

Total Combustibles E = (B + C + D) = ________

% Methane in Combustibles = (B/E x 100) = ________

% Hydrogen in Combustibles = (C/E x 100) = ________

From Fig 2. Lower Combustible Limit (F) = ________

From Fig 3. Nose Limit - % Oxygen. (G) = ________

From Fig 4 Nose Limit - % Combustibles (H) = ________

From Fig 5 Upper Combustible Limit (I) = ________

7. Procedure:

a. Construct Explosibility Triangle from (F)(G)(H) & (I) [See Inset]

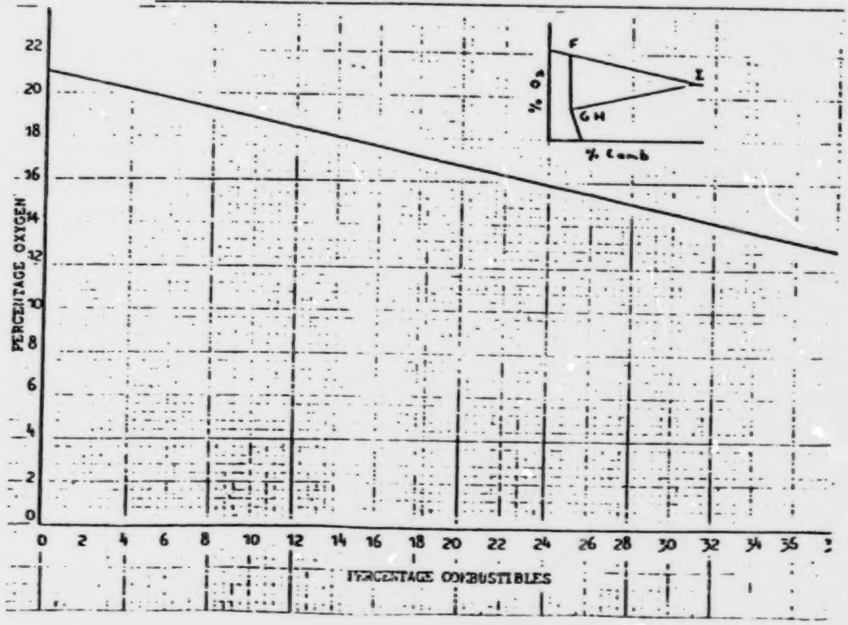
b. Draw Explosibility Boundary from Nose Limit to Combustible Axis such that extension of Line would pass through 20.93 % Oxygen

c. For Sample plot % Oxygen (A) against % Combustibles (E)

8. [illegible]:-

* EXPLOSIVE / EXPLOSIVE ON DILUTION / NON EXPLOSIVE

* Delete where NOT applicable



APPENDIX C

NITROGEN SUPPLIERS/CONTRACTORS

Nowsco Well Service Ltd.
Canadian Liqid Air Ltd.
Union Carbide Canada Ltd.

APPENDIX C - NITROGEN SUPPLIERS/CONTRACTORS

Company: NOWSCO WELL SERVICE LTD.

		MARITIMES	WEST
Address:		9 Pettipas Drive Burnside Industrial Park Dartmouth, Nova Scotia	900, 715 - 5 Avenue S.W. Calgary, Alberta
Telephone:		902-463-8136	403-261-2990
Contact Name:		Roy Mathew	Rod Muth
EVAPORATORS/PUMPERS*		1) SP-3000 Skid 2) SP-4500 Skid	SP-6000
On board storage	Sm^3		7000 m^3
Approximate liquid volume	Litres		10,500 L
Flowrate at outlet pressure of 15 bar		1) 85 m^3/min 2) 140 m^3/min	170 m^3/min
Maximum	m^3/min		
Minimum	m^3/min	1&2) 8 m^3/min	22 m^3/min
Number of units in fleet		Two	Ten
MOBILE STORAGE TANKS*		Skid Tanks	N_2 Bulkers
Capacity	Sm^3	5330 m^3	17,000 m^3
Approximate liquid volume	Litres	8000 L	25,500 L
Number of units in fleet		Twelve	Three
DELIVERY CAPABILITY FROM OWN SOURCES		(To Sydney N.S.)	(To Sparwood BC)
1. Maximum possible for 3 days	m^3/min	225 m^3/min	300 m^3/min
2. Maximum possible next 4 days	m^3/min	150	150
3. Maximum possible next 90 days	m^3/min	30	30

NOTE: Desired flow rates are:
1. >300 m^3/min
2. >150 m^3/min
3. > 30 m^3/min

* Please provide information for each size unit in fleet. Use more sheets if necessary.

APPENDIX C - NITROGEN SUPPLIERS/CONTRACTORS

Company: Canadian Liquid Air Ltd.

		MARITIMES	WEST
Address:			
Telephone:		SEE ATTACHED	
Contact Name:			
EVAPORATORS/PUMPERS*			
On board storage	Sm^3	NONE IN FLEET	NONE IN FLEET
Approximate liquid volume	Litres		
Flowrate at outlet pressure of 15 bar		SEE NOTES	
Maximum	m^3/min		
Minimum	m^3/min		
Number of units in fleet			
MOBILE STORAGE TANKS* $20^{Te} - 23^{Te}$		16,900 m^3 - 29,400 m^3 DEPENDING ON VARIOUS PROVINCIAL GOVT ROAD RESTRICTIONS	SAME
Capacity	Sm^3		
Approximate liquid volume	Litres	24,770 L - 28,440 L	SAME
Number of units in fleet		5 TRAILERS IN MARITIMES ADDITIONAL 25 TRAILERS AVAILABLE IN EAST	11 N_2 TRAILERS (#1)
DELIVERY CAPABILITY FROM OWN SOURCES		(To Sydney N.S.)	(To Sparwood BC)
1. Maximum possible for 3 days	m^3/min	88 m^3/ min.	60 m^3/min. (#4)
2. Maximum possible next 4 days	m^3/min	50 - 60 m^3/min.	40 m^3/min
3. Maximum possible next 90 days	m^3/min	30 - 35 m^3/min.	25 - 30 m^3/min.

NOTE: Desired flow rates are:
1. >300 m^3/min
2. >150 m^3/min
3. > 30 m^3/min

* Please provide information for each size unit in fleet. Use more sheets if necessary.

CANADIAN LIQUID AIR LTD.

3004-54th Ave., S.E., Calgary, Alta. T2C 0A7 (P.O. Box 1328)
Tel. (403) 279-2677 Telex: 03-821946

#1. We have 11 trailers currently in N_2 service in Western Canada, and an additional 11 trailers in Oxygen service which can be switched over to N_2 service.

#2. In the East, we have six (6) railcars, each with carrying capacity of 42,240 m^3 of N_2, which can be put into service for such an application.

#3. Canadian Liquid Air has a long-term contractual working relationship with Nowsco Well Service Ltd., spanning over the past 9 to 10 years, where the two companies work together to provide a complete package of N_2 availability and compression/vaporization capabilities to our customers.

#4. The delivery capabilities listed here could be improved upon, if enough forewarning is given, and if Canadian Liquid Air were to become contractually obligated to service the N_2 requirements of a particular mine.

LIQUID AIR IN CANADA

HEAD OFFICE: 1155 Sherbrooke Street West, Montreal, Quebec H3A 1H8

WESTERN REGION OFFICE: Suite 202 South, Heritage Square, 8500 McLeod Trail
Calgary, Alberta T2H 2N1

BRANCHES:
O'Connel Drive, Cornerbrook, Nfld. A2H 6E3
Kenmount Road, St. Johns Nfld A1B 3N4
520 Windmill Road, Darmouth, N.S. B2Y 3Z7
P.O. Box 428, Sydney, N.S. B1P 6H2
John Street, Moncton, N.B. E1C 8J1
4930 Hochelaga Street, Montreal, Quebec H1V 1E7
2230 ouest boulevard Charest, Quebec, Quebec G1N 2G3
135 boulevard Jacques Cartier, Sherbrooke Quebec J1H 5K5
535 rue Quebec, Sept-Iles, Quebec G4R 1K1
200 rue Cormier, Tracy, Quebec J3R 1V2
1700 Steeles Avenue East, Bramalea, Ontario L6T 1A6
850 Burlington Street East, Hamilton, Ontario L8L 4K3
1150 York Street, London, Ontario N5W 5G9
Building G, 866 Philip Street, Sarnia, Ontario N7T 1Z6
1194 Kelley Lake Road, Sudbury, Ontario P3E 4S6
Cameron & Waterloo Streets, Thunder Bay, Ontario P7C 4V5
1650 Field Street, Winnipeg, Manitoba R3E 3H8
Fourth Avenue & Winnipeg Street, Regina, Sask. S4P 3A2
501 Avenue M South, Saskatoon, Sask. S7M 2K8
3004 - 54th Avenue S.E. Calgary, Alberta T2C 0A7
10020 - 56th Avenue, Edmonton, Alberta T6E 5Z2
532 Slater Road, Cranbrook, B.C. V1C 4H7
No 1 - 444 MacAlpine Cres. Fort McMurray, Alberta T9H 4B1
P.O. Box 158, Kitimat, B.C. V8C 2G7
2565 McCulloch Road, Nanaimo B.C. V9S 4M9
8390 Manitoba Street, Vancouver B.C. V5X 3A7
60 Dallas Road, Victoria, B.C. V8V 1A2

APPENDIX C - NITROGEN SUPPLIERS/CONTRACTORS

Company: Union Carbide Canada Limited, Linde Division

123 Eglinton Avenue East, TORONTO, Ontario, M4P 1J3 (416)487-1311

		MARITIMES			WEST		
Address:		2525 Jean Battiste des Champs Blvd Lachine, Quebec			S. 311, 8925 - 51 Ave Edmonton, Alberta		
Telephone:		(514)636-4640			(403)468-5054		
Contact Name:		Deitmar Leorick			Dennis Jones		
EVAPORATORS/PUMPERS*		MPU-500	MPU-230	MPU-7341	MPU-540	MPU-230	MPU-7342
On board storage	Sm^3	5158	5158	4437	5158	5158	4437
Approximate liquid volume	Litres	9092	9028	7824	9092	9028	7824
Flowrate at outlet pressure of 15 bar							
Maximum	m^3/min	231	111	9	277	111	9
Minimum	m^3/min	23	8	9	23	8	9
Number of units in fleet		One	One	One	One	One	One
MOBILE STORAGE TANKS*							
Capacity	Sm^3	19,500	28,000			16,900	
Approximate liquid volume	Litres	28,500	40,000			24,770	
Number of units in fleet		Seven	Two			Five	
DELIVERY CAPABILITY FROM OWN SOURCES		(To Sydney N.S.)*			(To Sparwood BC)**		
1. Maximum possible for 3 days	m^3/min	35			60		
2. Maximum possible next 4 days	m^3/min	70			120		
3. Maximum possible next 90 days	m^3/min	70			120		

NOTE: Desired flow rates are:
1. $>300\ m^3/min$
2. $>150\ m^3/min$
3. $>30\ m^3/min$

*After Initial Transport Time 52 Hours
**After Initial Transport Time 12 Hours
Note: After 48 Hours Fleet can be increased by drawing on affiliated suppliers

* Please provide information for each size unit in fleet. Use more sheets if necessary.

APPENDIX D

CASE STUDY QUICK REFERENCE TABLE

CASE STUDY QUICK REFERENCE TABLE

Characteristic		Case Study Reference Number																						
		1	2	3	4	5	6¹	6²	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
1 COUNTRY	North America	x																	x	x	x	x	x	
	Western Europe				x	x	x	x	x	x	x					x	x	x						
	Eastern Europe		x	x									x	x	x									
	Other											x												x
2 TYPE OF MINE	Longwall	x	x	x		x	x	x	x	x	x		x	x	x	x	x	x						x
	Room-and-Pillar											x							x	x	x	x	x	
	Other				x																			
3. FIRE TYPE	Open	x	x	x	x	x		x			x		x	x						x				
	Inaccessible						x		x	x		x			x	x	x	x	x		x	x	x	x
4 FIRE LOCATION	Production Area	x	x	x		x	x	x	x	x	x		x	x		x	x	x						x
	Other Area				x							x			x				x	x	x	x	x	
5 METHANE	High	x	x	x		x			x	x	x		x	x	x			x						x
	Not High				x		x	x				x				x	x		x	x	x	x	x	
6 APPLICATION	Emergency	x	x	x	x	x		x	x	x	x		x	x	x	x	x	x	x	x	x	x	x	x
	Non-emergency						x					x												
7 INERTING MEDIUM	GCP		x	x								x	x	x	x									
	CO_2	x			x														x			x	x	
	N_2					x	x	x	x	x	x					x	x	x		x	x			x

APPENDIX E

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